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UNDERCUT BLOCK-CAVING METHOD OF MINING IN WESTERN COPPER MINES

BY

E. D. GARDNER

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE—BUREAU OF MINES

UNDERCUT BLOCK-CAVING METHOD OF MINING IN WESTERN COPPER MINES[1]

By E. D. Gardner[2]

INTRODUCTION

This paper is a progress report on the study of mining methods and costs in the western United States and summarizes that part of the work pertaining to the undercut block-caving method of mining.

Ore is mined on a large scale exclusively by the undercut block-caving method at the following six mines in the western United States: The Morenci branch of the Phelps Dodge Corporation, Morenci, Ariz.; the Ray mines of the Nevada Consolidated Copper Co. at Ray, Ariz.; the Inspiration Consolidated Copper Co. at Inspiration, Ariz.; the Miamo Copper Co. at Miami, Ariz.; the Ruth mine of the Nevada Consolidated Copper Co. at Ruth, Nev.; and the Consolidated Coppermines Corporation at Kimberly, Nev. The undercut block-caving method is also used at the mine of the Andes Copper Mining Co. at Potrerillos, Chile.[3]

At two mines—the Old Dominion Co. at Globe, Ariz., and the Copper Queen branch of the Phelps Dodge Corporation at Bisbee, Ariz.—caving methods are employed for obtaining a part of the production. The Cananea Consolidated Copper Co. at Cananea, Sonora, is using a caving system, and the Chino division of the Nevada Consolidated Copper Co. at Santa Ritan, Mex., is preparing a block of ground to be mined by this method. The Colorado Fuel & Iron Co. is starting the mining of an iron ore body at Sunrise, Wyo., by a caving method, and the Riverside Cement Co. is ready to mine a body of limestone at Riverside, Calif., by this system.

A caving method different in principle from that used at the mines listed above is employed at the gold mine of the Alaska-Juneau Gold Mining Co. at Juneau, Alaska.

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[1] - The Bureau of Mines will welcome reprinting of this article provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6350."

[2] - Supervising Engineer, Southwest Experiment Station, U. S. Bureau of Mines.

[3] - Greninger, I. L., Mine Development and Underground Construction of Andes Copper Mining Co. at Potrerillos, Chile: Trans. Am. Inst. of Min. and Met. Eng., 1929, p. 144.

Information Circulars describing the practices in detail at the Morenci,[4] Ray,[5] Inspiration,[6] Old Dominion,[7] Copper Queen,[8] Cananea,[9] and Alaska-Juneau[10] mines have been published by the bureau. Individual papers on some of the other mines will also be published. The current practice at the Miami Copper Co. has been described recently by F. W. Maclennan[11]

This paper summarizes, correlates, and compares the practices at the mines which have been described in detail. In addition to the published reports on individual mines, considerable data have been obtained from other mines visited and studied during this investigation.

ACKNOWLEDGMENTS

C. H. Johnson, assistant mining engineer, and W. O. Vanderburg, associate mining engineer, of the Bureau of Mines assisted in the collection of data contained in this paper. Mr. Johnson also prepared the original drawings pertaining to the practices at the Ray, Inspiration, and Copper Queen mines.

HISTORY OF DEVELOPMENT OF UNDERDUT BLOCK-CAVING METHOD

The undercut block-caving method may be defined as one in which blocks or sections of the ore body are undercut in some approrpriate manner; the ore breaks by its own weight and is then drawn from control chutes through grizzlies into storage or transfer raises, or directly into cars for transportation to the surface. The blocks are generally but not always cut off or weakened at the boundaries.

Block caving into incline chutes was started at the Ohio Copper mine in 1906.[12] The idea for this method was conceived from the sublevel caving system used in eastern iron mines and from the chute-caving method used at the Utah Copper mine for providing ore for a test mill and for weakening the ore body and capping preparatory to steam-shovel operations.

A good recovery of ore could have been obtained by the sublevel method, but it would have been necessary to shovel a large percentage of the ore into chutes, the cost of which would have made the method too costly for the low-grade copper deposits. The chute-caving method used at the Utah Copper Co. was satisfactory for the purpose for which it was intended; but there was little control of the ore, and the raises were spaced too far apart. Had operations continued long enough there would no doubt have been considerable loss of ore.

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[4] - Mosier, McHenry, and Sherman, Gerald, Mining Practices at Morenci Branch, Phelps Dodge Corporation, Morenci, Ariz.: Information Circular 6107.

[5] - Thomas, Robert W., Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz.: Information Circular 6167.

[6] - Stoddard, Alfred C., Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.: Information Circular 6169.

[7] - Shoemaker, A. H., Mining Methods at the Old Dominion Mine, Globe, Ariz.: Information Circular 6237.

[8] - Information Circular, Bureau of Mines, unpublished.

[9] - Catron, William, Mining Methods, Practices, and Costs of the Cananea Consolidated Copper Co., Sonora, Mexico: Information Circular 6247.

[10] - Bradley, P. R., Mining Methods and Costs, Alaska-Juneau Gold Mining Co., Juneau, Alaska: Information Circular 6186.

[11] - Miami Copper Co. Method of Mining Copper Ore: Technical Publication 314, Am. Inst. of Min. and Met. Eng.

[12] - McDonald, Felix, and Johnson, J. L., History of the Development of the Under-Caving Method of Mining: Meeting of the Arizona Chapter of the American Mining Congress, Oct. 10, 1927.

With this in mind the men in charge of the Ohio Copper Co. considered it possible to mine large blocks of homogeneous ore with good results by a caving method, provided the ore was properly undercut, the chutes evenly spaced, and the drawing of the ore controlled. The method was devised mainly by Felix McDonald, now mine superintendent of the Inspiration Copper Co. The first block developed was 100 by 100 feet in section and 30 feet thick. The vertical thickness was soon changed to 60 feet, as the deeper block made possible lower development cost and smaller dilution of ore from admixture of waste into the next block below. To prevent this dilution further, control chutes were placed as close as possible to the undercut level to minimize the penetration of waste rock into succeeding blocks. Higher blocks were not practical at the Ohio Copper Co. because of the presence of old filled stopes above.

The Inspiration Consolidated Copper Co. started development of a comparatively low-grade ore body in 1911 under Mr. McDonald's supervision. The system of mining used at the Ohio Copper Co. was adapted to the conditions existing at Inspiration. At the start, blocks 75 feet high were mined, as there was doubt concerning the feasibility of caving a 200 or 300 foot block of ore.

Contemporaneous with the work being done at the Inspiration mine, operations were begun at the Ray mine, now a division of the Nevada Consolidated Copper Co. The original method at Ray consisted of mining shrinkage stopes to the overburden above the ore, leaving alternate pillars which were cut off at the bottom and caved after the shrinkage stopes were completed. Both the shrinkage and pillar stopes were drawn at the same time. The present practice is an improvement on this first method.

Later the Miama Copper Co. adopted a caving method similar to that in use at the adjoining Inspiration property. The Nevada Consolidated introduced a caving method for mining the Ruth ore body at Ruth, Nev., in 1915. Beginning in 1922, the Morenci branch of the Phelps Dodge Corporation developed a caving system for mining a large body of low-grade ore at Morenci. The Copper Queen branch of the Phelps Dodge Corporation began mining low-grade ore bodies by a caving system in 1925. The Andes Copper Co. began mining by a caving system in 1926 and the Consolidated Coppermines in 1928. The Alaska-Juneau adopted a so-called caving method of mining in 1916, but the rock conditions are entirely different here from those at the mines discussed in this paper.

GEOLOGICAL FACTORS AFFECTING PRACTICABILITY OF CAVING SYSTEM OF MINING AND SELECTION OF METHOD

The following geological factors have an influence upon the mining of an ore body by the caving system and upon the selection of this method of mining:

1. Grade of ore.
2. Size and shape of ore body.
3. Thickness of capping or overburden.
4. Physical characteristics of the ore and capping.
5. Minerals in ore body.

Grade of Ore

As some dilution of ore with waste and some loss of ore always occur in mining ore bodies by a caving method, the grade of ore is an important factor to be considered before making a choice of this method. Where the expected loss of 10 or 15 per cent of the ore would be more important than the added cost of mining by some other method, a caving system would not be selected.

Size and Shape of Ore Body

Obviously, an ore body to be mined by an undercut caving method must be of sufficient size to justify the expense of running the haulage drifts and doing the preliminary development and undercutting work. Figure 1 shows a cross section of the Ray ore body of the Nevada Consolidated Copper Co. Figure 2 shows a plan of the Ray mine. The undercut block-caving method has been used on a large scale mainly in relatively flat-lying ore bodies. Ore bodies mined by this method should be fairly regular in outline, as fingers of ore extending into the country rock are lost and bodies of waste either inclosed within the ore body or extending into it from the edges must be mined. An exception to this occurs at the Copper Queen mine, where an ore body comparatively irregular in outline and partly with a steep dip is now being mined by this method.

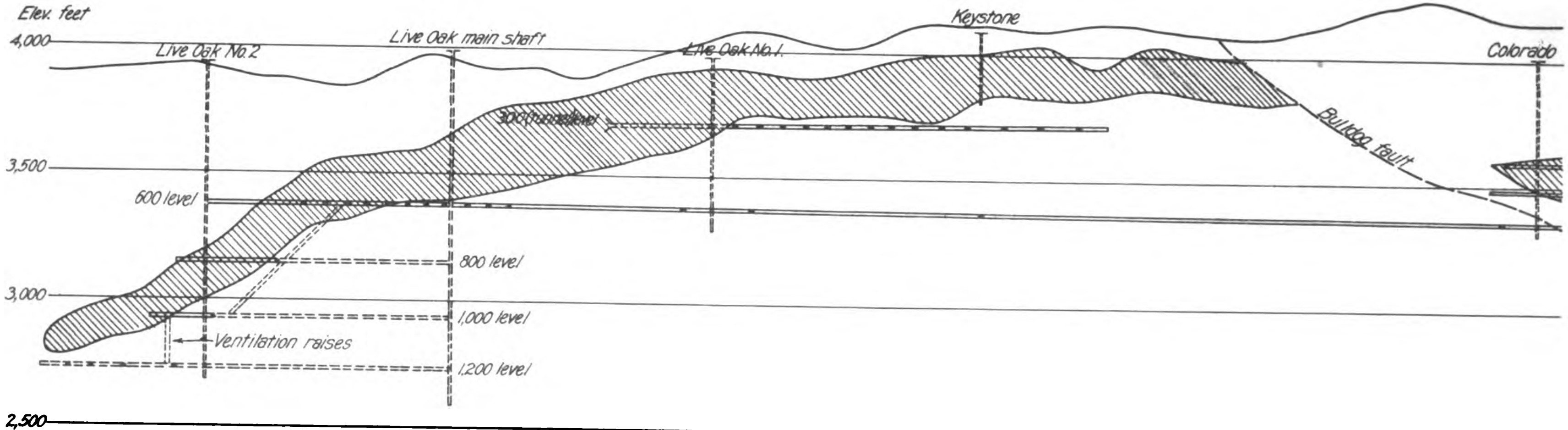
Thickness of Capping

Generally the thickness of the capping is the deciding factor, and climate and topography minor factors, in determining whether an ore body shall be mined by an open-cut or by a caving method, where either method can be used. Table 1 shows the thickness of capping and of ore at representative copper mines using open-cut and caving methods; also the ratio of stripping to ore mined at the open-cut mines is given.

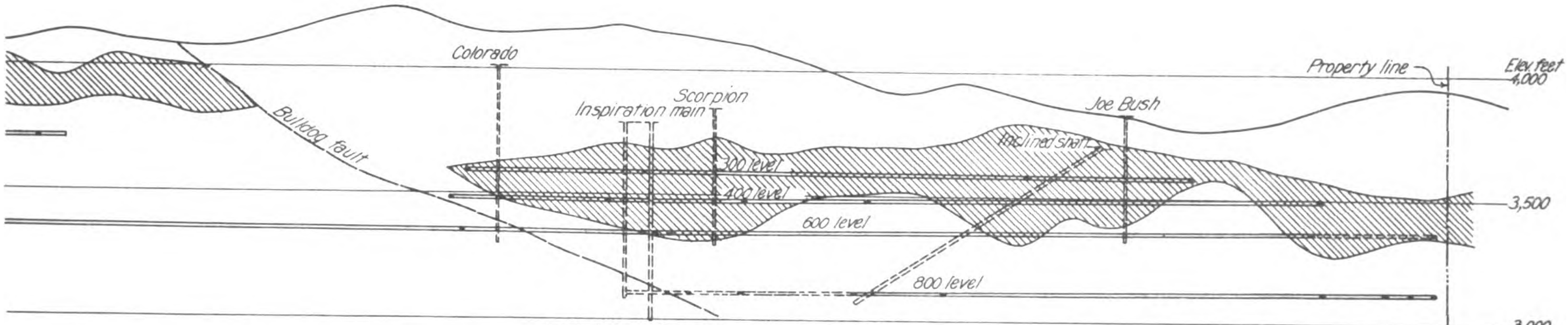
Although only a few hundred feet of capping overlies the ore at most mines where an undercut caving system of mining is used the Old Dominion is now beginning to mine a relatively small ore body on the 2,300-foot level by this method. Also caving operations are being carried on at the Cananea Consolidated Copper Co. at a depth of 1,500 feet from the surface.

Physical Characteristics of the Ore Body

A close study of the physical characteristics of the ore and of the minerals it contains will in most instances make it possible to predict how a mass of ore will act when it is undercut. Major planes of weakness should be considered in planning development work, and panels should be laid out in such a manner that adjustments of blocks due to undercutting would not cause stresses to be transmitted to active extraction workings. Whether the line of caving in blocks is parallel or across the principal set of minor fractures or jointing planes may have affect on mining operations in some cases.

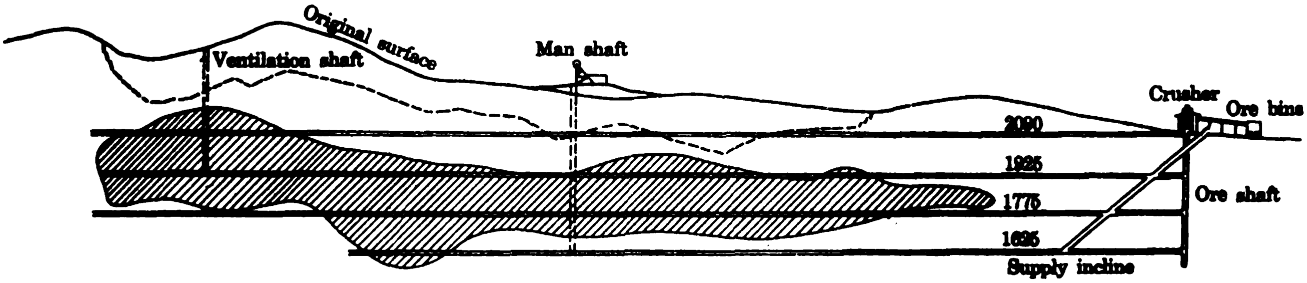


SOUTHWEST-NORTHEAST SECTION OF LIVE OAK ORE BODY

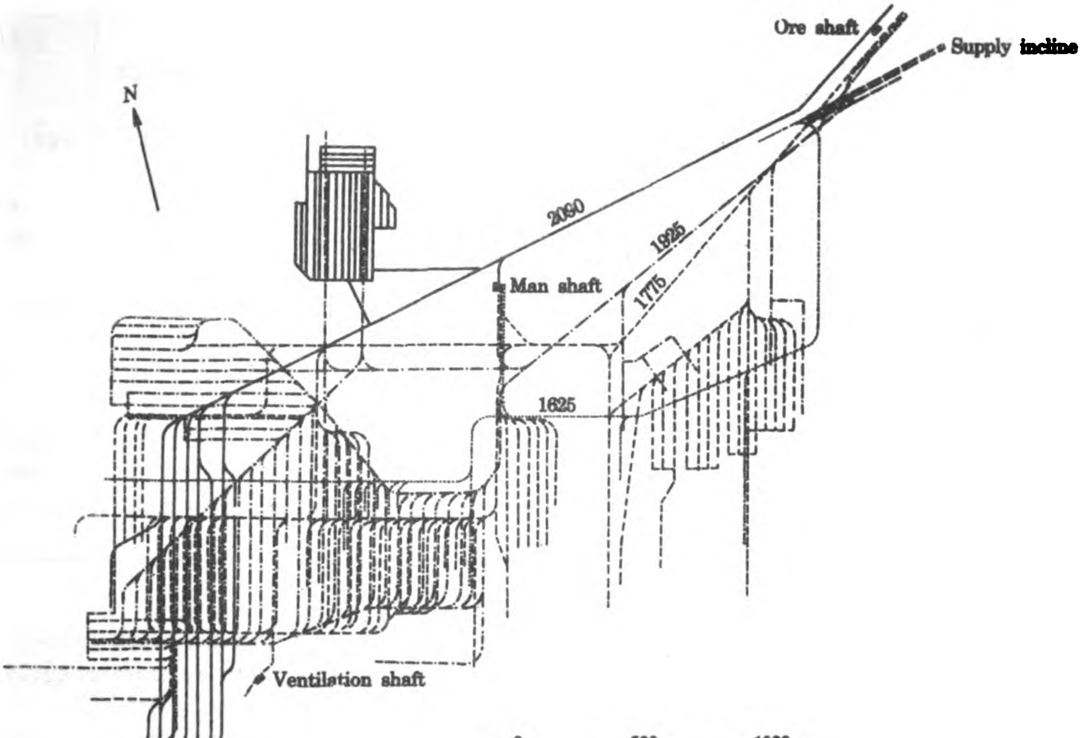


EAST-WEST SECTION OF INSPIRATION ORE BODY

Figure 1.- Sections of Live Oak and Inspiration ore bodies, showing shafts and haulage levels
Vertical and horizontal scales the same



SW.-NE. section



Plan

Figure 2.- Plan and cross section of ore body, Ray No. 2 mine

Table 1 - Relation of thickness of capping to ore at ten copper mines

Mine	Location	Grade of ore per cent copper	Average thickness of capping, ft.	Average thickness of ore, ft.	Ratio of stripping to ore mined
Caving method					
Ray (Nev. Cons.)	Ray, Ariz.	1.246c	250 (40 to 600)	120	
Inspiration	Miami, Ariz.	0.915	300 (0 to 500)	200	
Ruth (Nev. Cons.)	Ruth, Nev.	1.96	410 (110 to 540)	120	
Miami	Miami, Ariz.	0.83b	320 (250 to 500)	325	
Open-cut method					
Utah Copper	Bingham Canyon, Utah	0.994b	115	556	1.1:1 to date 1/2:1 total
Copper Flat (Nev. Cons.)	Ruth, Nev.	1.112d	190	224	1:1
New Cornelia (Calumet and Arizona)	Ajo, Ariz.	1.208b	0	250	—
Chino (Nev. Cons.)	Santa Rita, N. Mex.	1.49e	—	—	1.76:1
United Verde	Jerome, Ariz.	2.5 and 5.0	a	a	5.25:1 to date 2.41:1 total
Sacramento Hill (Copper Queen, Phelps Dodge)	Bisbee, Ariz.	2.28b	280	—	2.75:1 total

a - Ore a pipe-like mass extending to surface. Stripping mainly around ore mass. Open-cut method selected because of ore being on fire.

b - 1929.
c - 1928.
d - 1924.
e - 1923.

Ore bodies to be mined by an undercut block-caving method must be of such a structure that the ground will cave when the block is undercut. It is also desirable for the ore mass to contain sufficient fractures or be of such a texture that the ore will break in relatively small fragments in caving. Large, hard fragments of ore increase the cost of mining by this method and may make the system uneconomical in some cases.

Copper ore bodies suitable for mining by the undercut block-caving method are usually of the disseminated type. A rock structure that allows ready access of the mineral-bearing solutions through fractures is generally a favorable factor for mining the ore mass by an undercut caving system.

The ore mass in caving generally breaks to pre-existing planes of weakness or fractures. Examination of even the smallest fragment of caved ore will usually disclose that it has broken along fracture or jointing planes on practically all faces. This, of course, would not be true when the ore has undergone such alteration that it is more or less plastic.

Minerals in Ore Body

The mineral constituents of the ore are another important factor to be considered in mining by the caving method. Even distribution of the valuable minerals in the ore body is desirable, as selective mining can be practiced only to a very limited extent. Minerals that cause the ground to swell on exposure to air or moisture or minerals that cause the broken ore to stick in chutes or drawing raises are particularly undesirable in mining by this method. Excessive kaolinization or sericitization is likely to cause the ore to pipe on being drawn, which makes close spacing of draw points necessary. Secondary silicification may cement the fractures in the ore and otherwise increase its mechanical strength; this condition causes the ore to break into large pieces, making necessary large-size draw points and wide spacing of grizzly bars. Moreover, such ore requires that a large area be undercut before caving begins.

In mining by a caving method the last of the ore in a block may not be drawn until a year or more after it is broken; therefore, it is desirable that the ore body should not contain minerals that will oxidize or that will promote oxidation of the valuable minerals in cases where the ore mined is to be treated in concentrators. Such oxidation produces films on the sulphide minerals which interfere with flotation and increase the losses in the tailings. Some sulphides of iron oxidize rather readily and on oxidizing produce sulphuric acid and ferric sulphate, which attacks chalcocite and bornite. At the Ray mine a block of ore that had been undercut and caved was allowed to stand for about a year. Upon drawing it was found that part of the copper was in the acid-soluble form, which increased the losses in the tailings. This particular block had a relatively large amount of pyrite associated with the copper sulphide minerals.

At Morenci, Ray, and other low-grade copper mines in Arizona the sides of drifts in ore that have stood for two or three years show patches or coatings of copper sulphate sometimes 1/2 inch or more in thickness; this has been formed by the sulphating of the copper sulphides in the ore. In January, 1928, at the Ray mine a section of a stope was being drawn in which occurred a dike of decomposed basic rock that contained some disseminated iron sulphides which were undergoing alteration so fast that the odor of sulphurous acid was very disagreeable in the grizzly level.

Some evidence exists that ore consisting of sulphides of copper and iron with a high percentage of sulphur (over 45 per cent) has been ignited by the heat of friction developed by blocks of ore rubbing on each other when stopes have caved. Until it can be demonstrated that no possibility of fire exists due to caving of the ore, the caving method of mining should not be used in a massive-sulphide ore body.

Ore bodies to be mined by a caving method should be well drained. At some mines development work is done ahead of time to allow the ore to dry out before caving. Wet ore that contains clayey minerals is difficult to mine by a caving method. Besides sticking in chutes and raises, the ore packs after caving in the stopes, which makes the drawing operation expensive.

SURFACE SUBSIDENCE

In mining large ore bodies by a caving system of mining the surface is broken and subsides as the ore is drawn. Obviously, this method of mining can not be used when the ground surface must be preserved. Moreover, the method can not be used for mining smaller ore bodies at greater depths when it is undesirable to break the ground immediately above the stopes.

PRINCIPLES OF MINING BY UNDERCUT BLOCK-CAVING METHOD

The main features in mining by an undercut block-caving method are: (1) The breaking of the ore by the force of gravity, thereby eliminating drilling and blasting in stopes; and (2) drawing the ore under control through raises into haulage cars, thereby doing away with the necessity for shoveling.

The method is characterized by an intensive production of ore from a relatively small area which permits repetition of a number of simple operations. A thorough study of this work makes possible standardization of nearly all operations and efficient working of the mine.

Caving is induced by undercutting and cutting off the blocks from surrounding ground. The ore is drawn through openings uniformly spaced below the block.

By undercutting is meant the blasting and partial or complete removal of a layer of ore across the base of the column to be mined so as to permit the ore above to cave and break by gravity. The undercutting practice is not uniform at all mines, and the methods in use have been developed mainly to fit the particular conditions existing at each place.

Undercutting is supplemented by either cutting off the block entirely from the surrounding ground or weakening it along its boundaries so as to assist the caving action and confine it to the block mined.

For perfect drawing the draw points should be closely and regularly spaced, but a compromise must be made between close spacing, which increases ore recovery and reduces dilution, and wider spacing, with resulting economy in preparatory costs and operating repairs. The character of the ground is the controlling factor in drawing practice.

In the original Inspiration ore body and at the Miami and Ruth mines the ore breaks finely and tends to pack, and when drawn pipes vertically with little spread beyond the draw points. This condition makes it necessary to space the draw points as close together as economy and the necessity for maintaining supporting pillars will permit. At Morenci, Ray, and at the Copper Queen, where the ore is coarser and harder, the draw points are larger and spaced farther apart.

Dilution is caused by the irregular movement of ore and capping toward the drawing points forming pipes of waste which may reach a draw point far in advance of the top boundary of the ore, or by waste being drawn in from the sides or ends of a block. Dilution may also be caused by the general infiltration of fine capping down through coarsely-broken ore. Moreover, in all caving there is a gradual mixing of ore and capping as the ore is drawn down. If properly drawn a large part of the ore comes to the chutes clean. After dilution starts the proportion of waste increases and that of the ore decreases.

In mining a block of ore by a caving method three things must be constantly watched and controlled:

1. The mining must be done in such a manner that excessive weight does not develop on the grizzly level. This is generally controlled by the rate at which the ore is pulled.

2. The ore should be broken sufficiently in caving to be subsequently handled without further breaking. As a rule, the slower the drawing rate the more the ore is broken; also the slower the drawing rate the more weight is likely to develop on the grizzly level. Generally a balance is maintained between the two to get a maximum efficiency.

3. The ore must be drawn in such a manner as to get a maximum recovery and a minimum dilution with waste.

The point at which drawing of a block ceases depends upon the minimum grade of ore that can be handled and upon the value of ore that yields the best financial returns. If the plant capacity is large, drawing may continue until near the point of no profit. If not, to obtain adequate profits drawing must be stopped soon after dilution appears. The method is elastic in that by the sacrifice, for example, of 15 or 20 per cent of the ore, the grade can be maintained at very nearly its original figure.

MECHANICS OF CAVING

A difference of opinion exists as to the mechanics of caving. Observations from various mines under different conditions have not been correlated sufficiently to arrive at definite conclusions that would be applicable in all cases.

At Miami, with the present system of mining, it is considered that in each block the ore drops from the back of the arch into space made by undercutting or drawing until the caving line reaches the surface. Where this is the case it would appear that in caving only the tensile strength of the rock is overcome; any subsequent breaking of the rock fragments would occur while the blocks were traveling downward and being drawn from the draw points.

Where the ore is mined in long panels it is undercut and drawn, starting at one end in this case it is considered that some bending and shearing action occurs in the ore as it

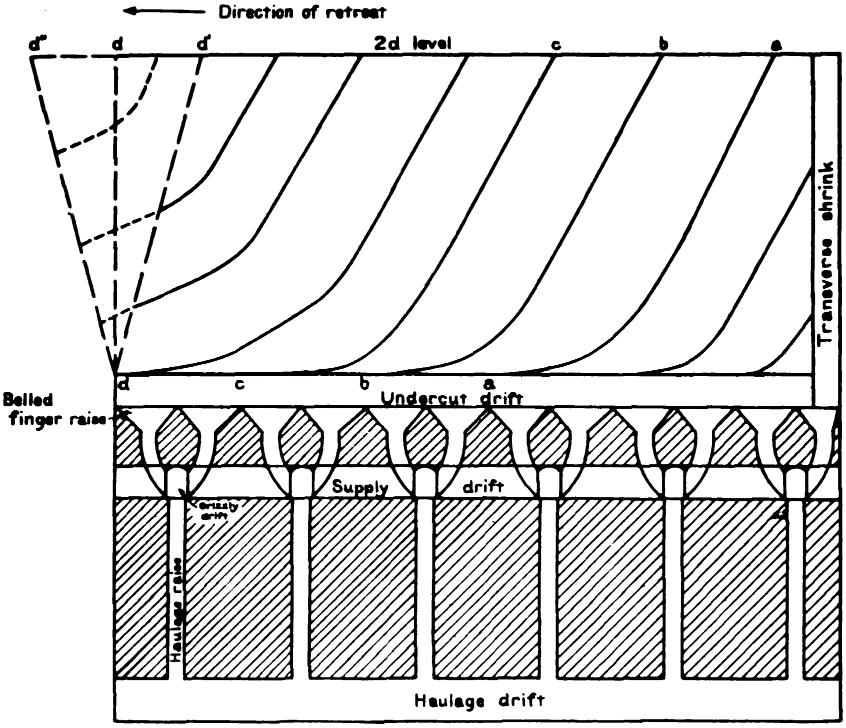


Figure 3.- Typical section showing progressive stages of caving; looking at right angles to line of retreat, Morenci mine

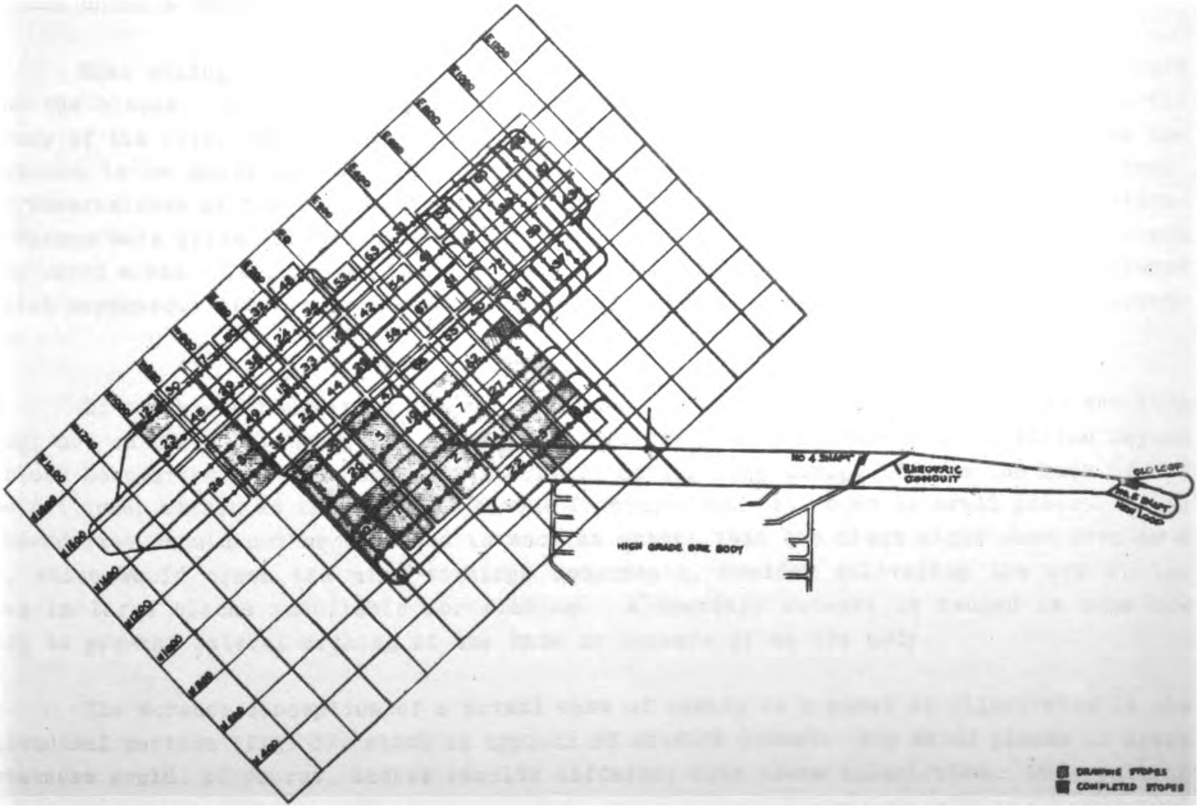


Figure 4.- Plan of 720-foot haulage level, Miami mine

caves. Considerable dust is made in caving, which probably indicates a grinding action of the mass as it travels downward.

At Morenci part of the ore is highly siliceous, and the fractures are more or less cemented. In this material some crushing or shearing would evidently be necessary to break up large blocks sufficiently to go through 16-inch grizzlies. The ore when first undercut breaks into large fragments that require considerable bulldozing at the grizzlies. When the undercutting has progressed in the proper manner, after this initial draw, the remainder of the ore is broken in caving to such a degree that most of it goes through the grizzlies. The first ore that caves into the space provided by the undercutting drops from the back, overcoming the tensile strength of the ore body only. As the ore drawn later is in smaller fragments, some additional crushing action is indicated.

At most of the other mines using a caving system the first ore caved usually is in larger blocks than that pulled later. At Ray, however, due to vertical ribs of hard ore in the deposit and to the method of undercutting the first ore drawn is in smaller fragments than the last pulled from the stope.

As stated previously, the slower the drawing rate the more the ore is broken up. This probably indicates that settling or readjustments in the caving mass cause large blocks to be broken up. This settling action is also probably responsible for weight developing in the workings below the drawing areas. Weight is more likely to develop under a large drawing area than under a small one.

When mining ore by the undercut block-caving method began the ore was not cut off around the blocks. At Inspiration it was first thought that in caving a block of ore the tendency of the outer edges of the block would be to cave outward. All early estimates included ore to an angle of 70° from the horizontal along all boundaries of the ore body. After observations of the action of several stopes the accuracy of this theory was questioned. Raises were driven along the sides of completed stopes and cross cuts at intervals run to the caved areas. This proved that the lines of the stope arched inward instead of outward as first supposed. After this discovery the blocks were cut off vertically along the boundaries.

If the block boundaries are not weakened the ore might arch to the center and stop caving; or, on the other hand, the caving might follow planes of weakness and extend beyond the block boundaries. It is desirable to retard caving long enough to have the back of the stope put under strain so that it will become fractured and cave down in small pieces. Also the boundaries should not be weakened to such an extent that the block might come down as a plug, which would crush the mine workings underneath, besides delivering the ore at the chutes in large blocks unsuitable for drawing. A boundary cut-off is nedded in some ore bodies to prevent lateral arching at the ends or corners of an ore body.

The Morenci conception of a normal case of caving in a panel is illustrated in the longitudinal section (fig. 3), which is typical of uniform ground. Any major planes or areas of weakness would, of course, induce results different from those illustrated. Undercutting and drawing begins at the right end of the panel and retreats to the left. The actual cutting is delineated in the figure. Caving is initiated along a rather flat inclined brow which often approximates a slope of 30°. This steepens against the caved ore to perhaps 60°C. With a sufficient time interval this brow stabilizes in an almost vertical position at the extreme edge of the undercut. It may (1) project slightly over the caved ore, (2) stand exactly perpendicular, or (3) even extend at a small angle beyond the basal undercut (d', d, d'' in figure).

In all mines using an undercut caving method the maintenance of extraction openings is an important part of the operations. Some rock on being exposed to air or moisture swells or readily spalls. Other rock may be so weak structurally that it will not support itself even for a short time in small openings. In addition, some of the extraction workings tend to collapse as the equilibrium of an area is disturbed by removing the support beneath blocks in undercutting. This tendency to collapse is termed "taking weight" by miners. Why the workings should take weight, as is the case in many instances, is not definitely understood, although it can be largely prevented in most cases by following the right procedure of undercutting and drawing, ascertained by experience at each mine. Apparently the taking of weight is caused by dynamic rather than static loads. For weight to develop in a working, some movement of ground must take place. This movement is generally along well-defined planes of weakness or due to readjustment of blocks in the rock, except where the whole mass is fractured to such an extent that a movement analogous to flow occurs. Also in a clayey ore body the rock may tend to be plastic; in this case increased weight over a working would tend to destroy any equilibrium that existed and cause the working to collapse. Increased weight may be caused by the abutments of an arch or the fulcrum of a cantilever, by unbroken pillars in an undercutting area, or by the dead weight of a mass of caved ore whether in one block or broken up.

In mining by an undercut block-caving method weight may develop on the grizzly level under the undercutting or drawing areas for occasionally as much as 300 feet ahead of the undercutting operations. Where excessive weight develops immediately ahead of the caving or undercutting area it would appear that it was due to an unbroken mass of ore acting as a cantilever. The taking of weight at relatively long distances ahead of the undercutting or on the haulage level is probably due to pressure being transmitted betwen planes of weakness. This weight ahead of the undercutting can be relieved by running transverse shrinkage stopes across a panel, which in effect cuts it into smaller blocks.

At Ruth, where the ground is heavy and structurally weak, repairs are continually needed in the haulage levels. Drifts at this mine under areas that have been worked out and thereby a part of the overlying load removed are easier to hold open than in virgin ground.

SIZE OF BLOCKS AND PANELS

Two general plans for stoping are followed: (1) Panels along or across the ore body, or (2) individual stopes of more or less definite length. The ore bodies are usually laid out in panels to accommodate the haulage system. The panels are then mined as a continuous stope or cut into blocks, depending upon the conditions existing at each mine. The sizes of the panels and of blocks used at a number of mines are shown in Table 2. The width of the panels varies from 75 to 200 feet. The length is up to 1,200 feet, depending upon the size and shape of the ore body.

Table 2 - Spacing of development work preparatory to caving at eight copper mines.

	Morenci	Ray	Inspiration		Miami	Andes Copper	Old Dominion	Ruth	Copper Queen
			"Chute set control"	Grizzly control					
Normal daily tonnage	4,500	12,000	(1)	(1)	18,000	6,600	—	3,500	(2) 2,500
Distance apart of haulage drifts feet	56	50	85	90	150	109.3	—	75	100
Distance apart of gathering raises do.	28	25	25	25	50	20.5 and 30.7	40	25	40
Distance from haulage level to grizzly level do.	50	40	39	45 to 60	100	39.4	—	50	50 to 100
Inclination of gathering raises degrees	75	55+	50	50	55	54	—	50 to 65	60
Distance apart of grizzly drifts or raises feet	28	25	42-1/2	25	50	54.6	20	12-1/2	40
Distance apart of draw points along grizzly drifts or in a drift do.	21 & 14	25	25	16	25	27.3	10	12-1/2	20
Plan area served by each draw point do.	14x18.66	—	—	12.5 x 16	12.5 x 12.5	13.7 x 13.7	10x10	12.5 x 12.5	20 x 20
Area served by each draw point square feet	261.3	312.5	132.8	200	156.25	186	100	156.25	400
Distance undercut level above grizzly level feet	20	—	30	18	30	32.8	—	10	16
Distance apart undercut drifts or raises do.	18.6 & 14	12.5	25	11 to 14	25	27.3	10	12.5	20
Width of panels do.	112	200	170	180	150	164	125	(3)75	100
Length of panels do.	up to 600	up to 600	up to 1,000	up to 1,000	up to 1,200	656	350	up to 500	up to 200
Width of blocks do.	—	200	170	180	150	—	—	—	100
Length of blocks do.	—	150 to 200	(4)	(4)	150	—	—	—	125 av.
Height of block do.	150 to 270 av. 164	100 to 300	125 to 225	125 to 225	320	(5)330	100	40 to 200 av. 135	60 to 250

1 - Total of 20, both controls. 2 - Caving only. 3 - Varies according to conditions of ore. 4 - Depends upon character of ore and mining conditions. 5 - Sulphide ore body.

In an individual stope method more places can be worked at once, which gives flexibility to drawing operations and less congestion in haulage laterals. Also more selective mining can be done should two classes of ore, such as leaching and concentrating ore, be mined from the same deposit. Better control of drawing is possible in a small block where the ore is drawn down slowly...

In a panel stope control of the drawing is more difficult, as different quantities of ore are drawn from the chutes on different days to keep the right angle of contact between the ore and capping as the stope advances. Also a hung-up chute or one that is closed on account of an early appearance of waste is less serious in a block with a flat angle of contact.

With the panel system fewer haulage laterals need to be kept open and fewer grizzlies drawn at the same time, resulting in a lower maintenance cost. Also, as the panel is worked from one end to the other fewer end cutoff workings are needed. The panel system has an advantage in hard ground with moderate lifts, as the shearing and bending of the ore as it breaks from the solid at one end tend to break the mass into smaller fragments than would be the case if it were drawn evenly in blocks.

In comparing the advantages of mining the ore by individual stopes or in panels, F. W. Maclennan states:[13]

> In the earliest caving methods at Miami, the entire width of the ore body was undercut and caved, beginning at one end and retreating along the length of the ore body, closely followed by ore drawing and endeavoring to maintain a plane of contact between this broken ore and barren capping with a dip away from the direction of retreat at an angle of from 40° to 60° from the horizontal. Caving over such a considerable width, 500 or 600 feet, resulted in excessive weight being thrown on the extraction openings, with correspondingly heavy maintenance costs and interference with orderly ore drawing. Later experience indicates that a width of 150 feet caused satisfactory caving in Miami ore with moderate maintenance costs, and it became standard practice to cave and draw alternate panels 150 feet wide across the entire ore body. A year or so later, when the waste rock which had settled into these original panels had consolidated, the pillar panels were caved and drawn back across the ore body between them. This method was well adapted to the comparatively low lifts then in use but it was felt that it would not be satisfactory with the high lifts of 300 feet or more which were essential to low-cost mining in the low-grade ore body.
>
> In order to maintain an angle of contact between the broken ore and capping of 40° to 60° in an ore column 300 feet, it would be necessary to maintain a length of 300 feet or more of extraction levels open and in good condition for ore drawing along the panel, and it is likely that this would not always be possible. Excessive weight would almost certainly interfere with orderly ore drawing and in some cases to such an extent that it would be necessary, in order to maintain the required tonnage, to drop back and cave virgin ore, abandoning the broken ore overlying the caved workings. The temptation to do this is always present in the retreating panel system and results in lost ore and increased cost per ton for the reduced tonnage. In this system, employing the inclined plane of contact between broken ore and waste, and with a great thickness of ore, there would be great danger of dilution. There would also be a likelihood that damaging weight would be thrown on the area just ahead of the undercutting by the overhanging cantilever

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13 - Maclennan, F. W., Miami Copper Co. Method of Mining Low-Grade Ore Body: Tech. Pub. 314, Am. Inst. Min. and Met. Eng., 1930.

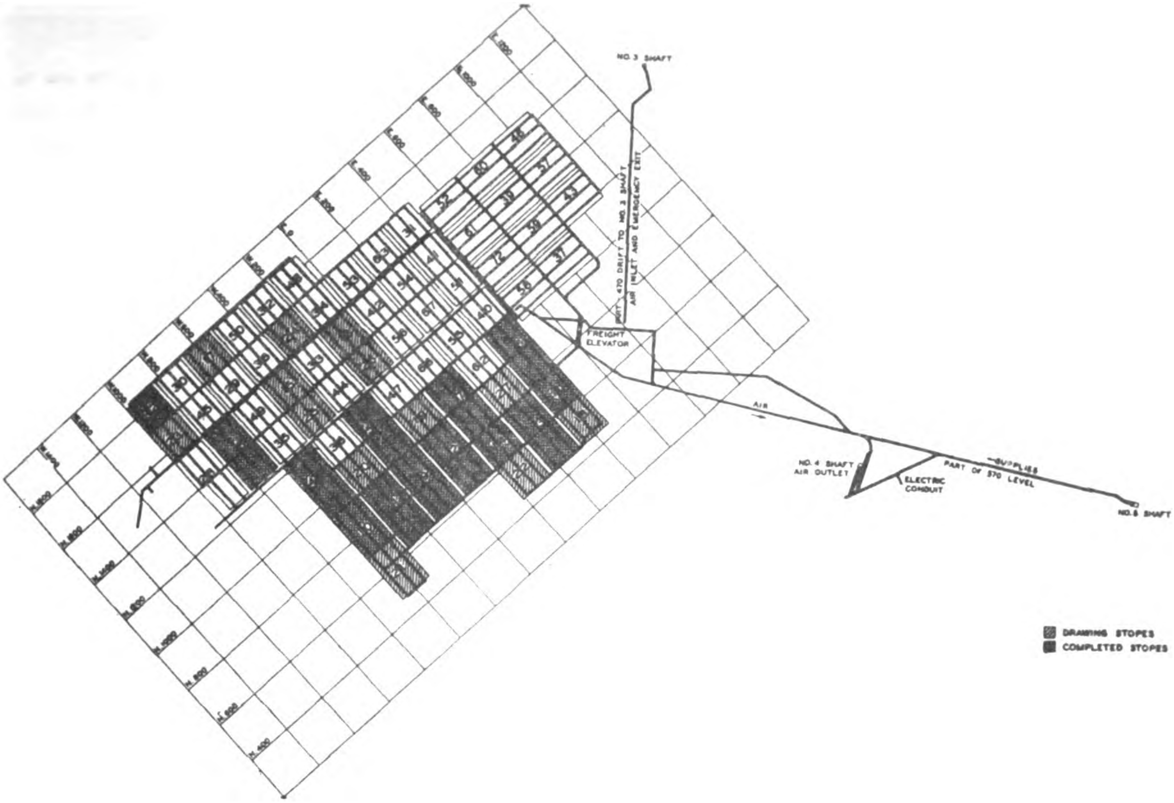

Figure 5 – Plan of 620 and 635 foot grizzly levels, Miami mine

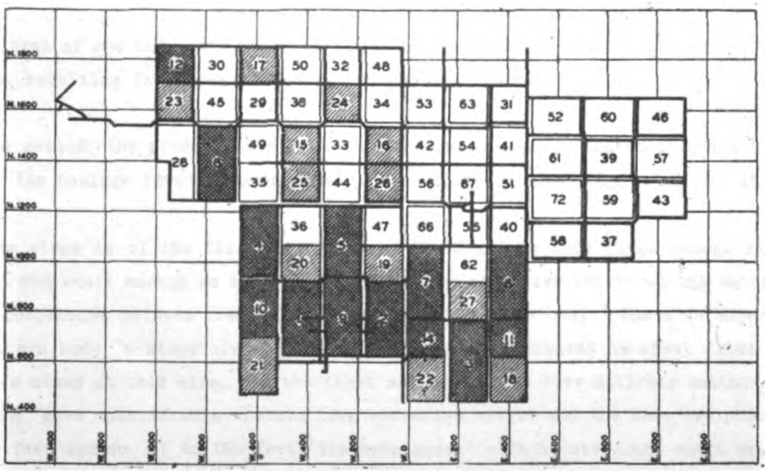

Figure 6.– Plan of 510-foot boundary caving level, Miami mine

brow of ore which has been undercut but not yet caved. When this happens it is frequently necessary to cut off this cantilever of ore at the fulcrum point by putting up a narrow shrinkage stope across the panel. If this happened often, this panel method would evolve itself into a series of individual stopes but without the advantage of having been systematized and proper control arranged for in advance.

It was decided that a system of mining utilizing individual stopes was best adapted to mining in one lift an ore body whose thickness was 300 feet or more. Advantages claimed for this method compared with any method in which caving progresses across the ore body are in addition to those indicated above:

1. The area to be caved is confined to a definite area and is surrounded on all four sides by solid ground in the case of the original stopes and by consolidated fill in part or in whole in the case of pillar stopes, compared with an indefinite area, one end of which is recently caved capping the other end recently undercut ore both in motion, and the sides solid ground in the case of the pillar panels. Undoubtedly the support of the individual stope is much better, hence lower maintenance costs, and less interference of repair work with ore drawing, tending to better tonnage extraction with less dilution.

2. The order of mining of the stopes is laid out in such a manner that pillar stopes will not be mined until the waste fill along any boundary has been consolidated for several months. This fill becomes quite a substantial support during this period. The order of the stoping is shown in Figures 4, 5, and 6, which are plans of the 720-foot haulage level, 620-foot grizzly level, and 510-foot boundary caving level. The stopes are mined in the order in which they are numbered on these plans....................

3. In the individual stope method the ore is drawn down evenly over the entire area of the stope, resulting in a horizontal plane of contact between the broken ore and broken waste overlying, and while ore drawing with an inclined plane of contact between ore and waste is satisfactory for low lifts, there can be no question of the superiority of the horizontal plane of contact for high lifts. The inclined-plane method has been used exclusively in the Miami ore body with good results in low lifts since 1913.

4. In the same area of ore body, the stope method provides a greater number of working places and the work is more easily standardized, resulting in a greater tonnage production.

5. In the stope method, the production can be more conveniently distributed over the area to deliver the ore to the various drifts on the haulage level in economical proportions avoiding congestion and delays.

The area of the stope is of the first importance. It should be made large enough so that the ground will cave freely when undercut and small enough so that it will not throw excessive weight on the extraction openings below. In other words, it is a compromise between free caving and low maintenance cost. Based on experience with other caving methods in the Miami ore body, a stope area of 150 by 300 feet was selected as about right for the average ore. The first nine stopes were mined at this size, and the first eight of these were entirely satisfactory. Stope 9, which was the first pillar stope, gave considerable trouble from excessive weight and the next two pillar stopes were reduced in length, No. 10 to 225 feet and No. 11 to 150 feet. The development work in stope 12, which was an original stope in particularly weak ground, gave warning of trouble if it were opened out to full size, so it was reduced to an area of 150 by 150 feet; this size gave such favorable results both in the original and the pillar stopes that it was adopted as standard and the remaining stopes were laid out in this size, with the exception of 11 stopes at the east end of the ore body, which were made 150 by 200 to conform to the closer spacing of the haulage drifts, made necessary because the sills of these 11 stopes were at a lower elevation than the stopes to the west.

Width of Panels

The width of panels depends upon many factors, which in turn are governed principally by the size, shape, and physical characteristics of the ore body. Panels are usually laid out to fit the ore body and haulage system. A relatively small ore body would probably be mined in one block or panel.

A panel must be wide enough so that when the ore is undercut it will cave by its own weight but not wide enough to be the cause of excessive weight being transferred to the extraction openings. In weak ground, for instance, the ore caves more readily, but weight is more likely to develop on the openings below.

The width of panel is an important factor in laying out the development and mining system. At some mines existing levels have been utilized for haulage or grizzly levels. In this case the width of the panel must be made to fit the established vertical interval.

At Morenci the ore body is divided into stoping panels 112 feet wide, which extend more or less on the strike of the ore from one limit to the other. As mining proceeds from one end of the panel it can be followed by another stoping operation in an adjoining parallel panel after enough time has elapsed to allow the caved waste capping to settle in the finished stope.

At Ray, panels with the so-called sublevel caving system are 200 feet wide, mined in blocks 100 feet long. Two modifications of this general method, called the "motor" and "modified motor," are also used at Ray. In these methods the panels are 160 feet wide and the blocks usually 100 and 130 feet long, respectively. At Ruth, due to the heavy ground, the panels are usually 75 feet wide, but in firmer ground the width of the panels is increased.

The ore body at the Old Dominion, now being mined by the Morenci slide method, is 200 by 350 feet in cross section and has a thickness of 100 feet. Transverse shrinkage stopes are run to prevent overthrust settling on advanced development work, but the entire ore body will be considered as one stope. Each transfer raise serves four slides, or an area of 40 feet square (fig. 7).

Height of Lifts

The tendency in caving operations in the Southwest is toward higher lifts, thus reducing preparatory or development costs.

As stated before, the height of the first blocks mined by an undercut method was only 30 feet. This was soon increased to 60 feet. The first block at Insperation was 75 feet high. As experience has been gained, the lift has been steadily increased. In 1928 the average stoping lift at Morenci was 164 feet, while the next lower stoping horizon will be 270 feet. At Miami, blocks over 300 feet are being successfully caved. The curve in Figure 8[14] shows that the cost per ton of mining rapidly decreases as the thickness of the ore

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[14] - See footnote 13.

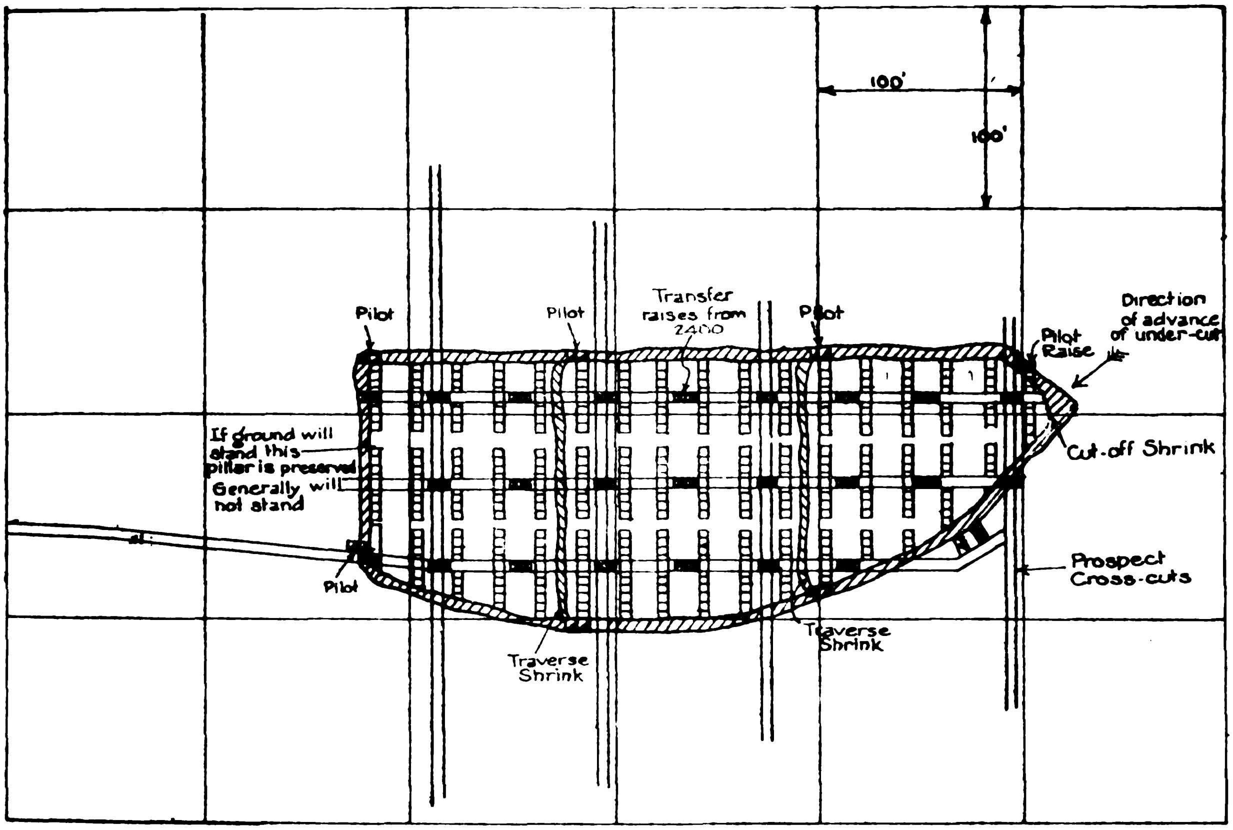
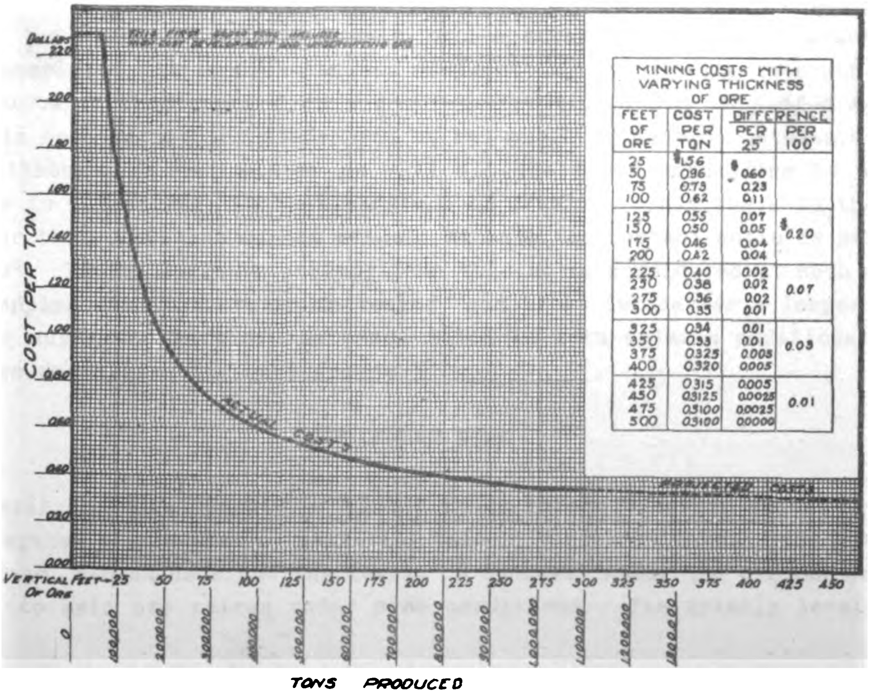


Figure 7.- Plan of caving method, Old Dominion mine



MINING COSTS WITH VARYING THICKNESS OF ORE

FEET OF ORE	COST PER TON	DIFFERENCE PER 25'	DIFFERENCE PER 100'
25	$1.56		
50	0.96	$0.60	
75	0.73	0.23	
100	0.62	0.11	
125	0.55	0.07	$0.20
150	0.50	0.05	
175	0.46	0.04	
200	0.42	0.04	
225	0.40	0.02	0.07
250	0.38	0.02	
275	0.36	0.02	
300	0.35	0.01	
325	0.34	0.01	0.03
350	0.33	0.01	
375	0.325	0.005	
400	0.320	0.005	
425	0.315	0.005	0.01
450	0.3125	0.0025	
475	0.3100	0.0025	
500	0.3100	0.0000	

Figure 8.- Relation of mining costs to thickness of ore, Miami mine

increases up to 100 feet high. The cost per ton continues to decrease up to a life of 300 feet, after which the curve is nearly flat.

An increase in the height of the ore column above 250 or 300 feet reduces the cost of stoping very little. It increases the total length of time required for the ore to reach the mill after being broken in the stope. This delay in drawing may be detrimental to certain ores that oxidize rapidly for concentration by gravity or flotation.

Angle of Contact Between Ore and Capping

In general, the angle of contact maintained between the top of the ore and the capping depends upon local conditions, such as the height of the lift, the size of the block, the necessity of protecting a stope from side dilution, the tendency of the workings under the drawing area to take weight, and the dilution of the ore by admixture of capping.

At some mines experience has shown that less side dilution is obtained if the ore adjoining completed stopes is drawn last. To obtain this result a relatively steep angle of contact between the ore and overlying capping is necessary.

A relatively steep angle of contact is used where the mechanical strength of the ore is low or if the ore has a tendency to be plastic due to excessive fracturing or a large proportion of clayey minerals. In this case less weight develops on the extraction working and the length of time necessary to keep it open is less.

With extremely steep angles of contact dilution of the ore by the overlying capping is increased. Usually the angle of contact at each mine is determined by experience to give a balance between loss caused by dilution and repair costs due to weight.

At Ray the blocks are drawn evenly, and a horizontal angle is maintained after undercutting is complete. At Inspiration the undercutting and drawing is begun at the side or corner of a block farthest away from a completed section. The rate of drawing depends upon conditions in each particular stope, and no set angle of contact between ore and waste is maintained, although as flat a draw as will fit the other conditions is desired. At Morenci the angle is 60° during the initial stage of drawing. This angle is then flattened to 50° or less for the completion of the stope. At Ruth the contact antle is kept as nearly as possible to 40°. If the angle is steeper than this it is considered at Ruth that the admixture of the capping with the ore is increased, and if it is flatter a longer time is required for mining any given part of the panel which in turn entails additional expense in maintaining raises and drifts in heavy ground.

DEVELOPMENT WORK

In general the development for mining by an undercut caving system consists of a haulage level, a system of transfer or gathering raises, a grizzly level, draw points, boundary cut-off workings, and an undercutting level. A sublevel under the grizzly level is used for tramming ore to main ore raises under some conditions. The grizzly level is usually used as the main working and ventilation level for the work above the haulage level; a service drift on the grizzly level may be run in advance of haulage raises.

Running boundary cut-off workings and doing the undercutting are usually classed as mining and the other work is called development.

The sequence of the preparatory operations is based mainly on convenience in disposing of the ore broken in stope preparation. All these operations are carried on at the same time, each process being sufficiently in advance of that following it to avoid interference. Moreover, the work is planned far enough ahead that none of it needs to be hurried for a stope to be ready for production.

Haulage Levels

As large tonnages are taken from relatively small areas an efficient haulage system is necessary. The loop system has an advantage in that all traffic under the stopes is in the same direction, but has a disadvantage that when a drift is closed, due to heavy ground, all traffic along the line is held up while repairs are made. Figures 4, 9, and 10 show the layout at the Miami, the Morenci, and the Ruth mines, respectively.

The general layout of the haulage level depends upon the size and shape of the ore body and the distance from the hoisting shafts or haulage adit. The distance apart of the main drifts or laterals is governed principally by the height of the grizzly or a transfer level above the haulage level, the distance apart of the grizzlies on the grizzly levels, and the minimum angle of inclination on which the ore will run freely in the raises. As shown in Table 2, the main haulage drifts or laterals are 50 to 150 feet apart.

Figure 11 shows the timbering and cross section of a haulage drift at Morenci, and Figure 12 shows the details of one at Ruth.

The ore is delivered to the haulage level through raises and loaded into cars through chutes. At some places the raises are placed such a distance apart that every third car of a train is under a chute so two or three cars can be loaded at a time. At the chutes pony sets are put in to provide space for the car loaders. The pony set and loading chute at the Inspiration mine are shown in Figure 13.

Transfer or Branch Raises

The length and type of a transfer or branch raise depends upon the distance between haulage drifts and the height of the grizzly level above the haulage drift, which height may or may not have been established before the ore was blocked out.

Unless the ore is mined in more than one lift it is usually desired to have the drawing level at the bottom of the ore body to make as complete extraction as practical and not to mine any waste. Where the bottom of the ore is inclined, as at Inspiration or Ruth, the length of the raises will vary in accordance with this condition.

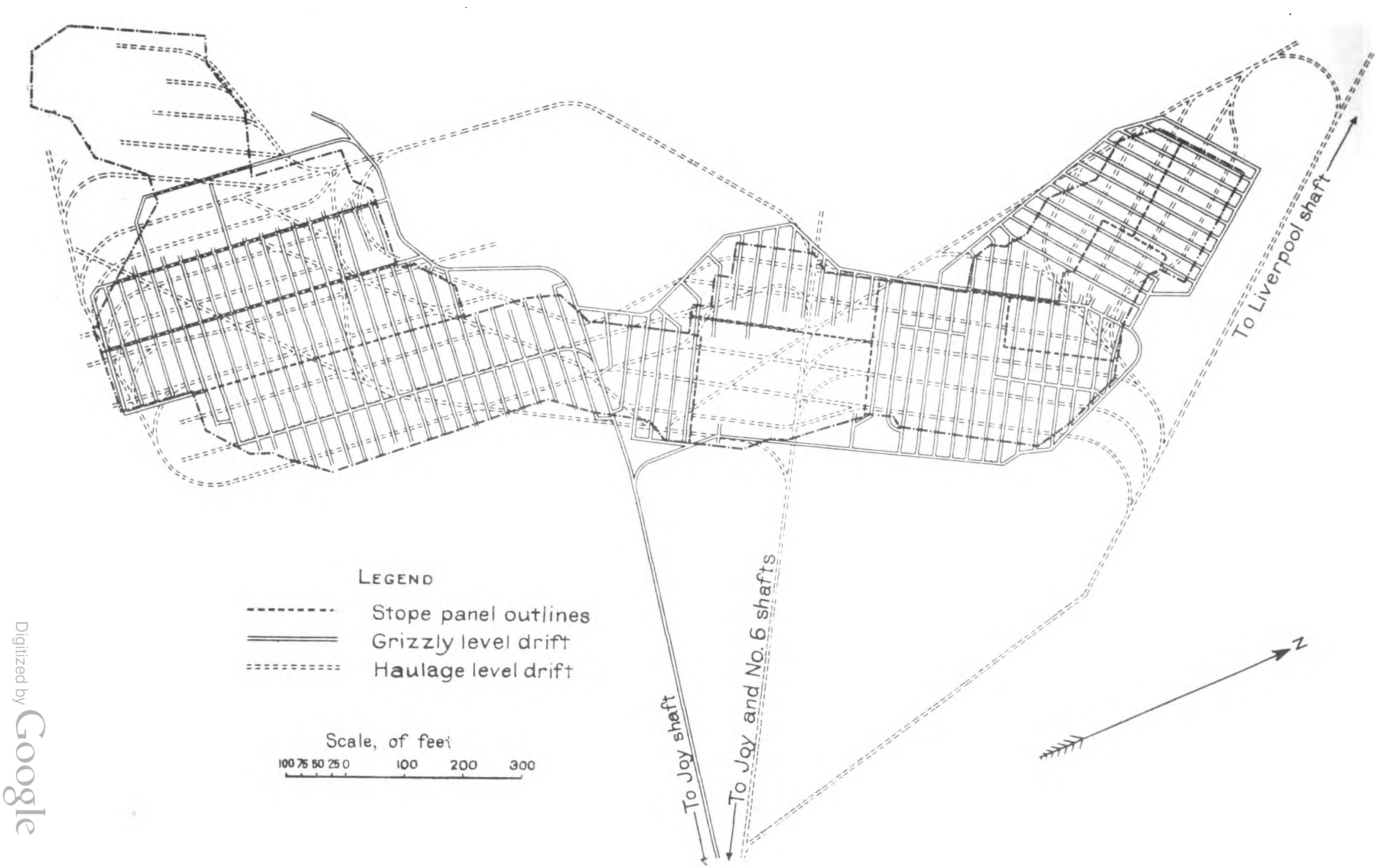


Figure 9.- Plan of grizzly and haulage levels, Morenci mine

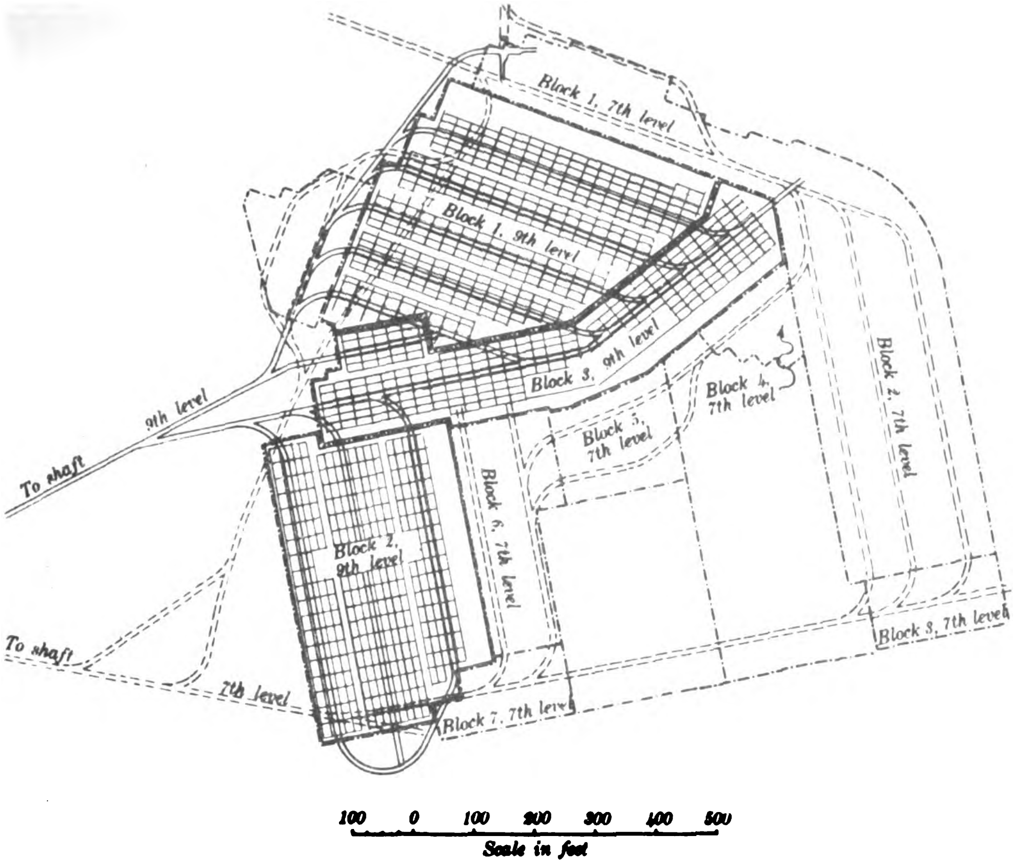
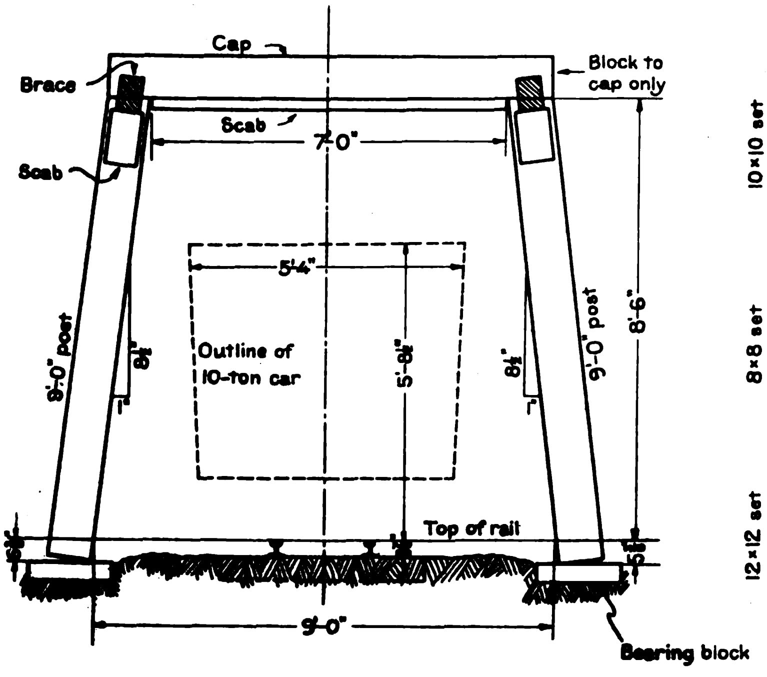


Figure 10.- Plan of seventh and ninth levels at Ruth mine, showing caving blocks



LIST OF TIMBER FOR 1 SET

Set	Pieces		NO.	DESCRIPTION	BD.FT.
10×10 set	2	Posts	39	10x10x9-0 Batter 1 End As Shown	150.0
	1	Cap	42	10x10x8-8 Sq Ends	72.2
	1	Scab	128	2x10x7-0 " "	11.7
	2	Scabs	44	2x8x1-0 " "	2.7
	2	Braces	63	6x8x4-2 " "	33.3
	2	Bearing Blocks	45	4x12x1-8 " "	13.3
				TOTAL BOARD FT.	283.2
8×8 set	2	Posts	46	8x8x9-0 Batter 1 End As Shown	96.0
	1	Cap	49	8x8x8-4 Sq. Ends	44.3
	1	Scab	11	2x8x7-0 " "	9.3
	2	Scabs	44	2x8x1-0 " "	2.7
	2	Braces	62	6x8x4-4 " "	34.6
	2	Bearing Blocks	45	4x12x1-8 " "	13.3
				TOTAL BOARD FT.	200.2
12×12 set	2	Posts	52	12x12x9-0 Batter 1 End As Shown	216.0
	1	Cap	56	12x12x9-0 Sq. Ends	108.0
	1	Scab	27	2x12x7-0 " "	14.0
	2	Scabs	44	2x8x1-0 " "	2.7
	2	Braces	61	6x8x4-0 " "	32.0
	2	Bearing Blocks	45	4x12x1-8 " "	13.3
				TOTAL BOARD FT.	386.0

Figure 11.- Details of standard haulage-drift set, fifth level, Morenci mine

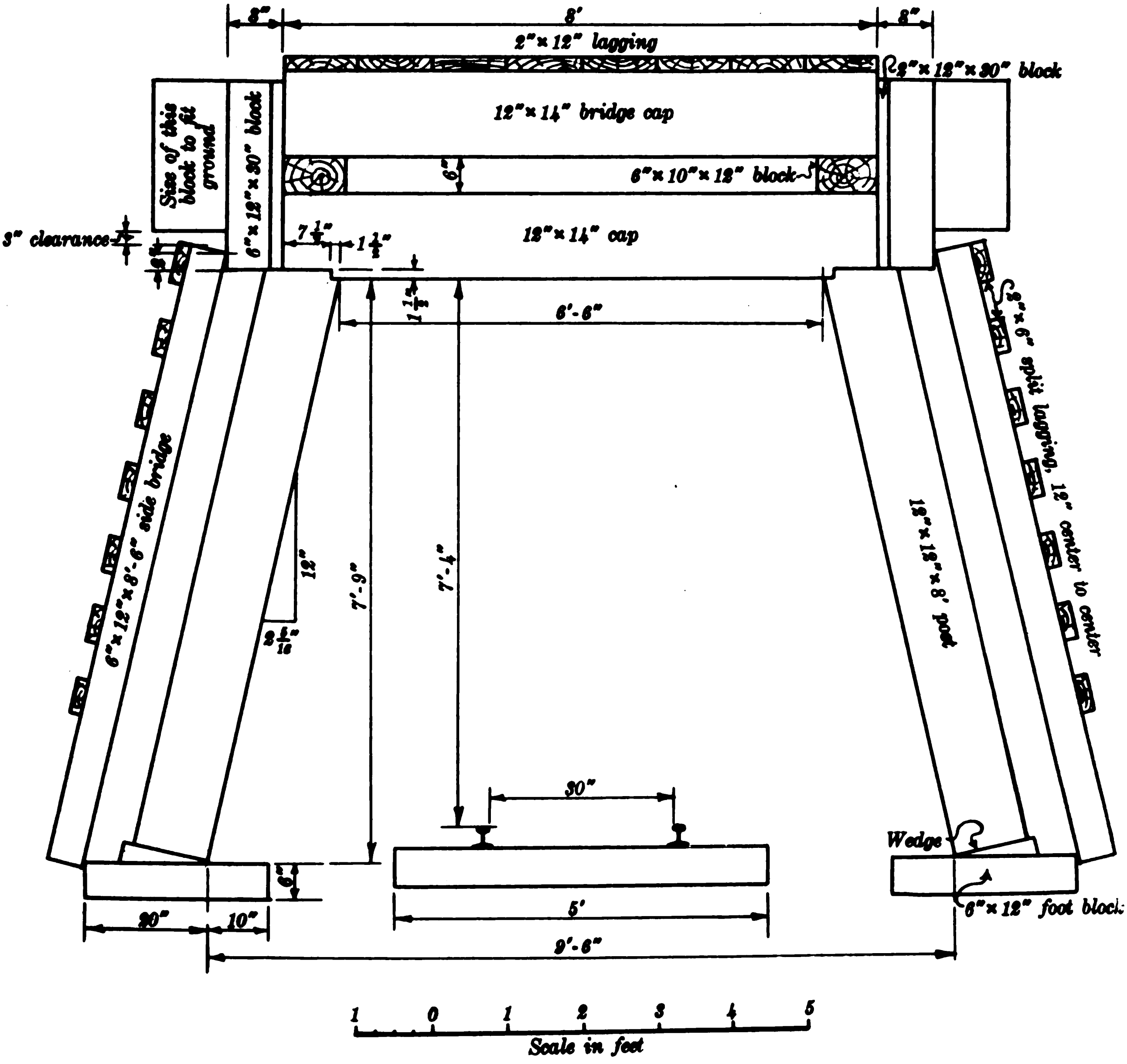


Figure 12. – Motor drift set in heavy ground, Ruth mine

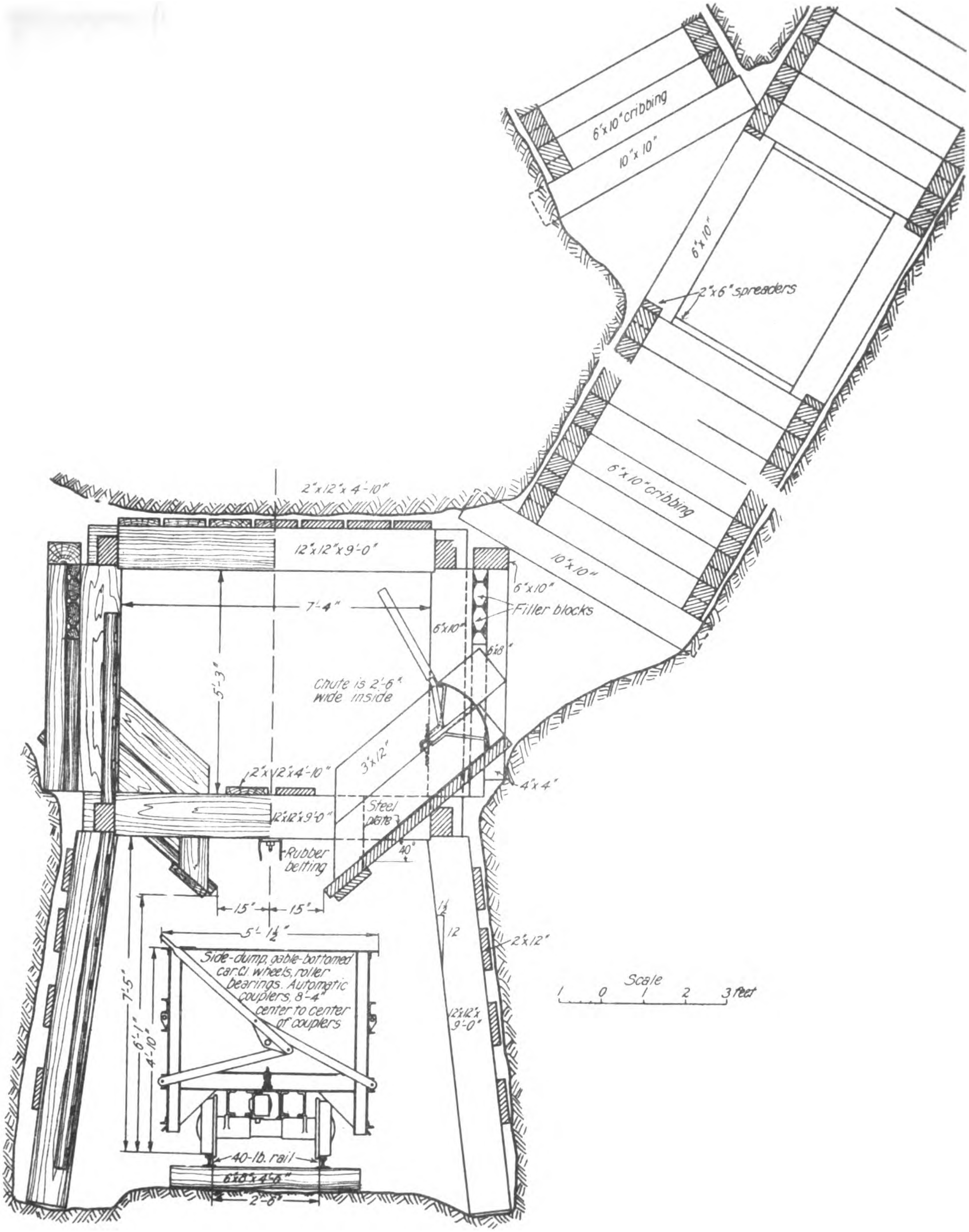


Figure 13.- Drift set, pony set, mine ore car, loading chute, and branched raise cribbing at Inspiration mine

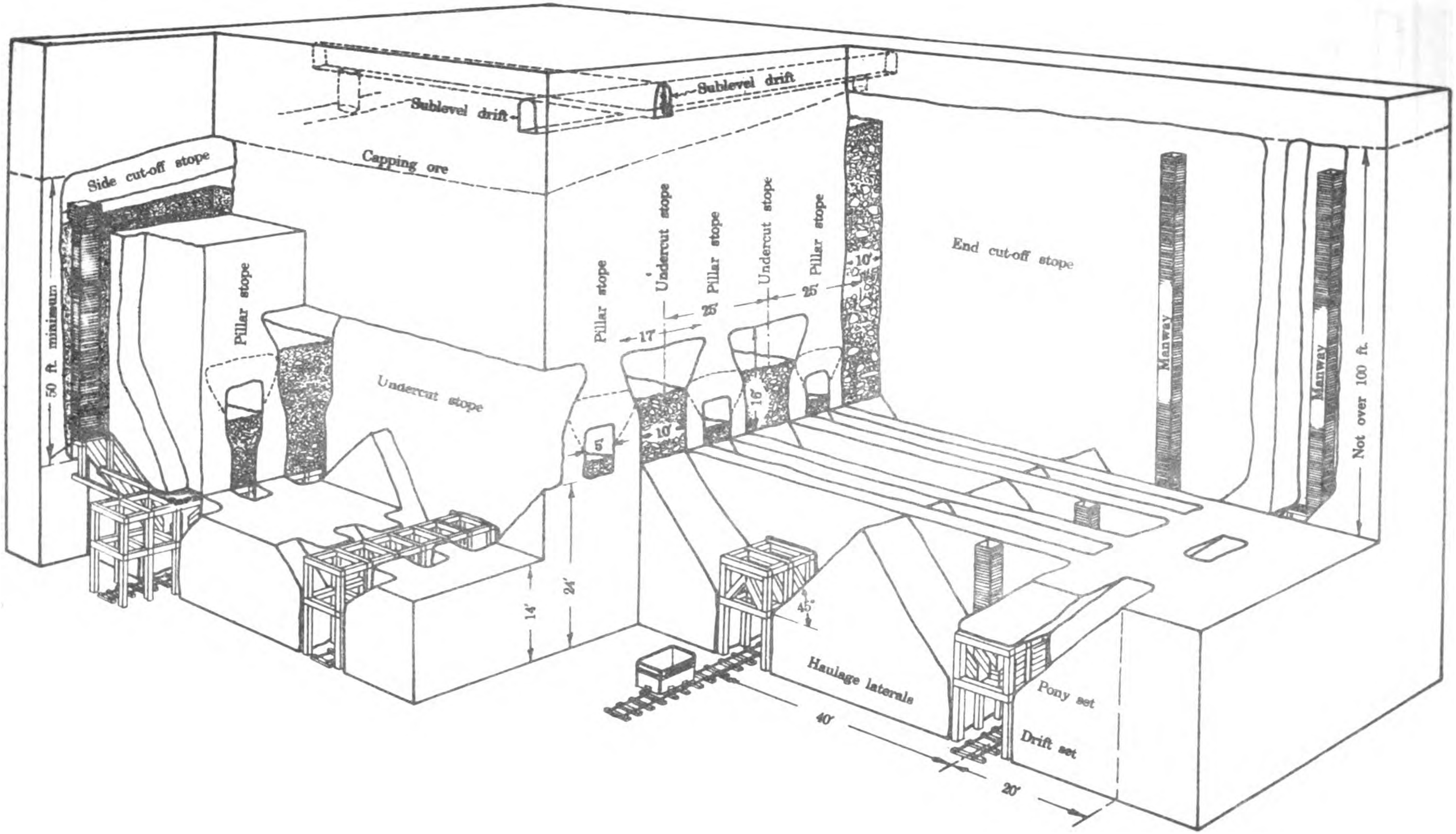


Figure 14.- Motor-haulage caving method, Ray mine

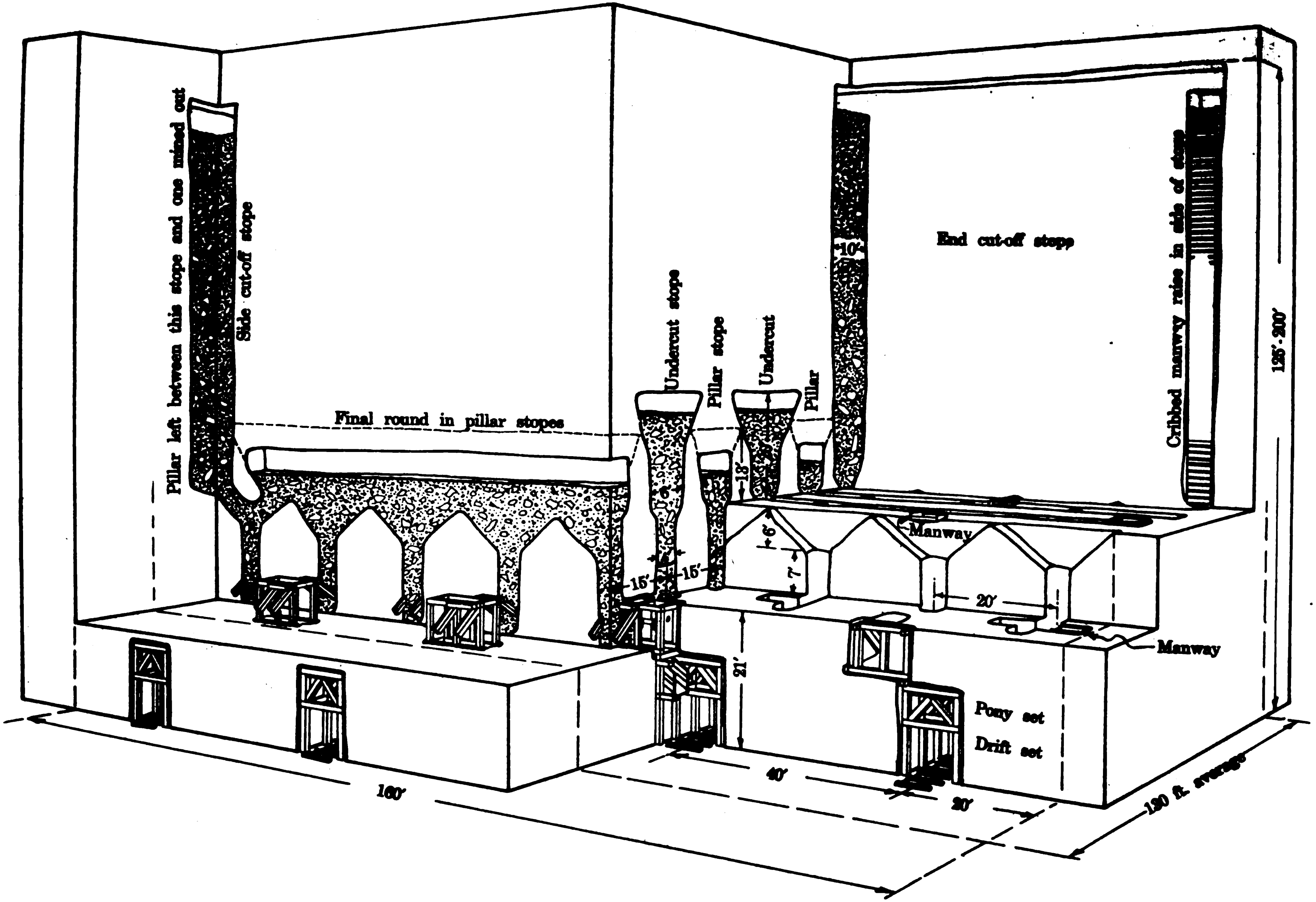


Figure 15.- Modified motor-haulage caving method, Ray mine

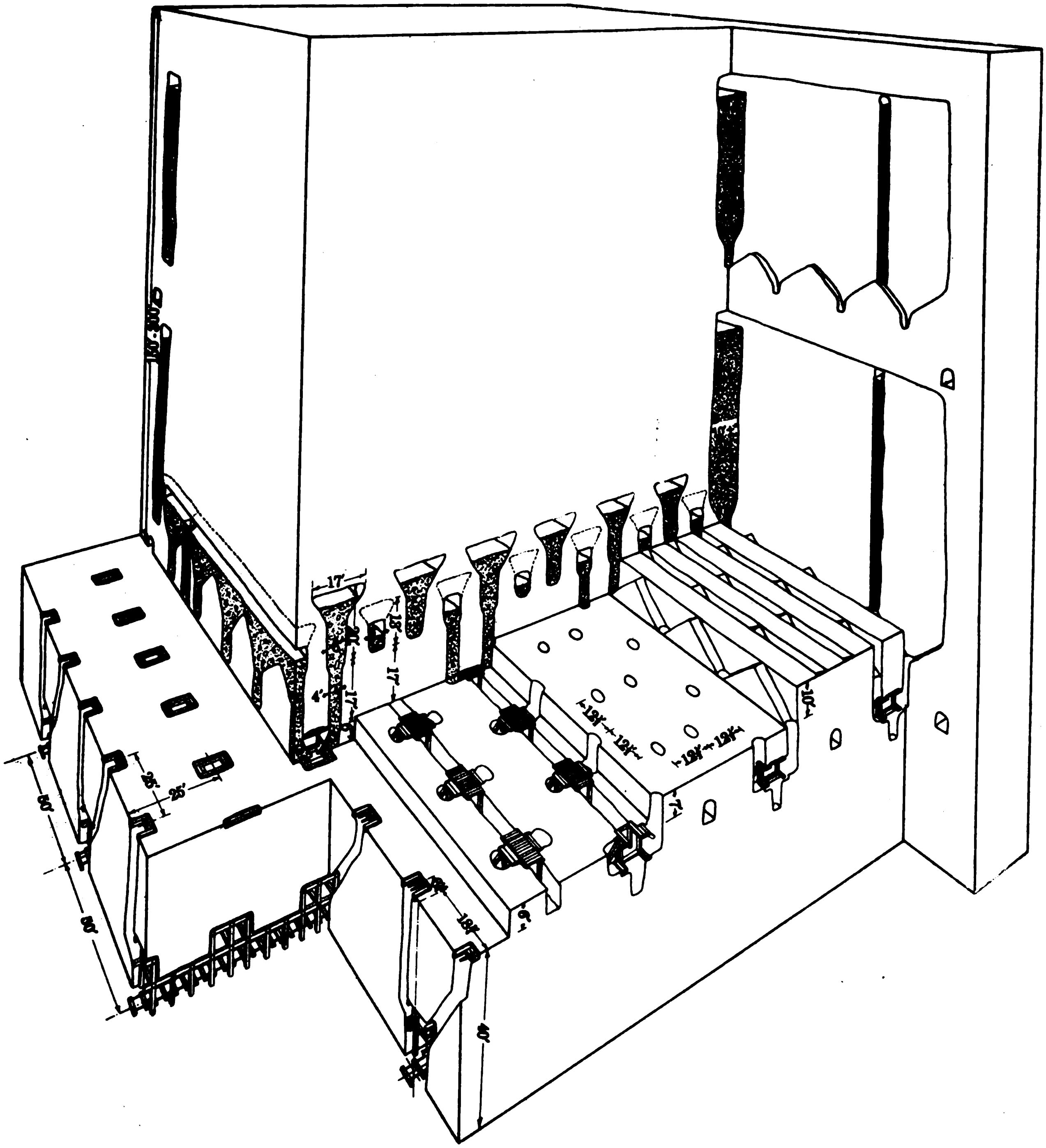


Figure 16.- Sublevel undercut caving method, Ray mine

TIMBER SCHEDULE

No.	Amt.	DESCRIPTION
321	8	12x12x3-6 Sq.Ends (2 invisible underpieces)
324	6	12x12x4-0 " "
322	12	12x12x5-0 " "
141	4	12x12x5-6 " "
68	8	12x12x6-0 " "
140	8	12x12x6-2-1/16 Beveled, See Detail
323	2	12x12x8-0 Sq. Ends
114	8	12x12x14-0 Sq. Ends
103	8	10x10x3-6 Sq. Ends
345	8	3x12x20-15/16 For 50 Rails
30	12	3x12x5-6 Sq. Ends
212	4	3x12x5-0 " "
12	4	2x12x12 Sq. Ends
16	8	2x12x1-9 " "
273	4	2x12x2-6 " "
29	4	2x12x3-6 " "
31	4	2x12x4-0 " "

Below are listed pieces not shown on this drawing but which may be required in abnormal conditions to substitute for certain pieces listed in above timber schedule.

344	-	10x10x5'-0" Sq. Ends
113	-	10x10x5'-6" " "
57	-	12x12x12'-0" " "
268	-	12x12x7'-0" " "
325	-	12x12x10'-0" " "

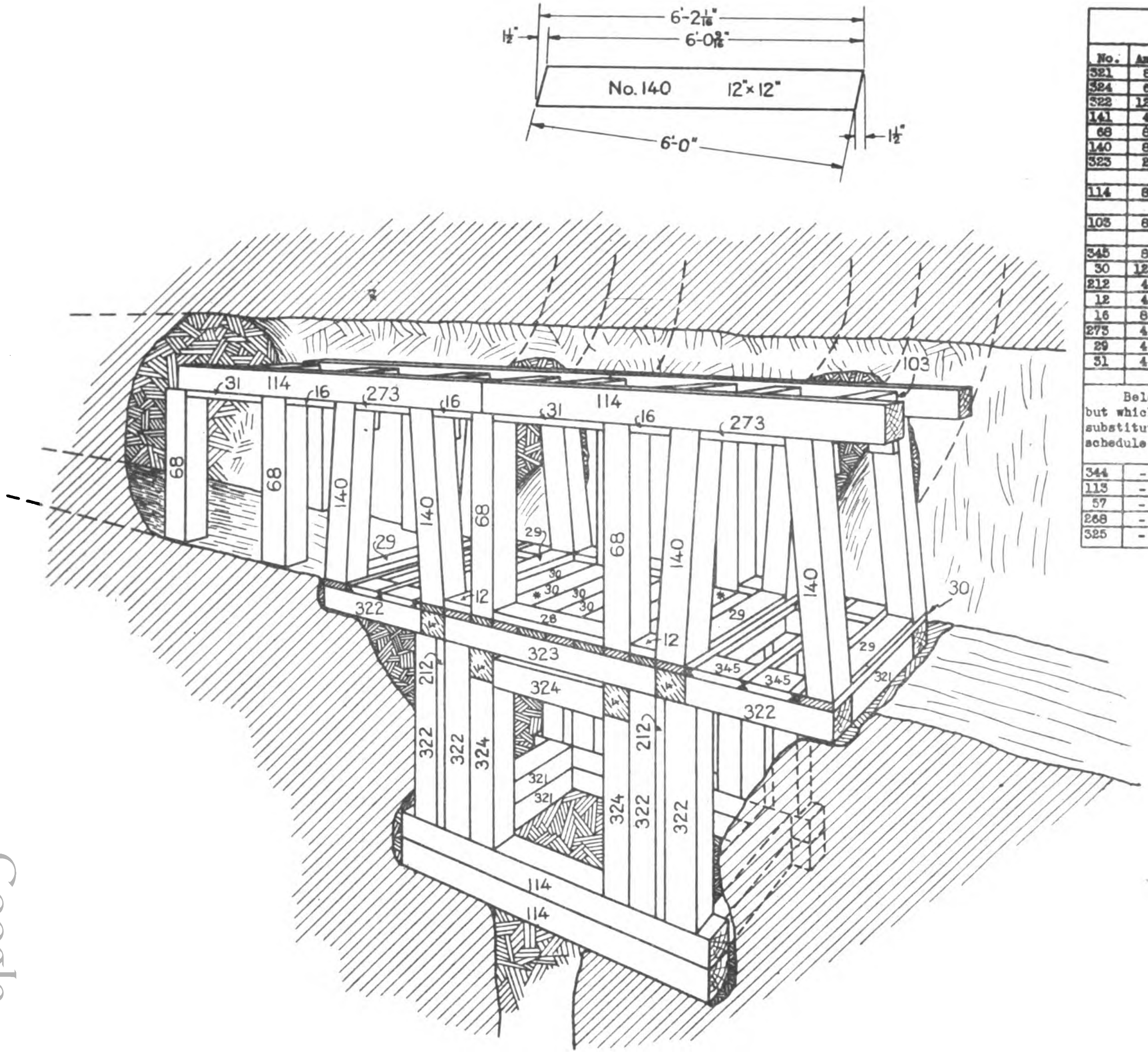
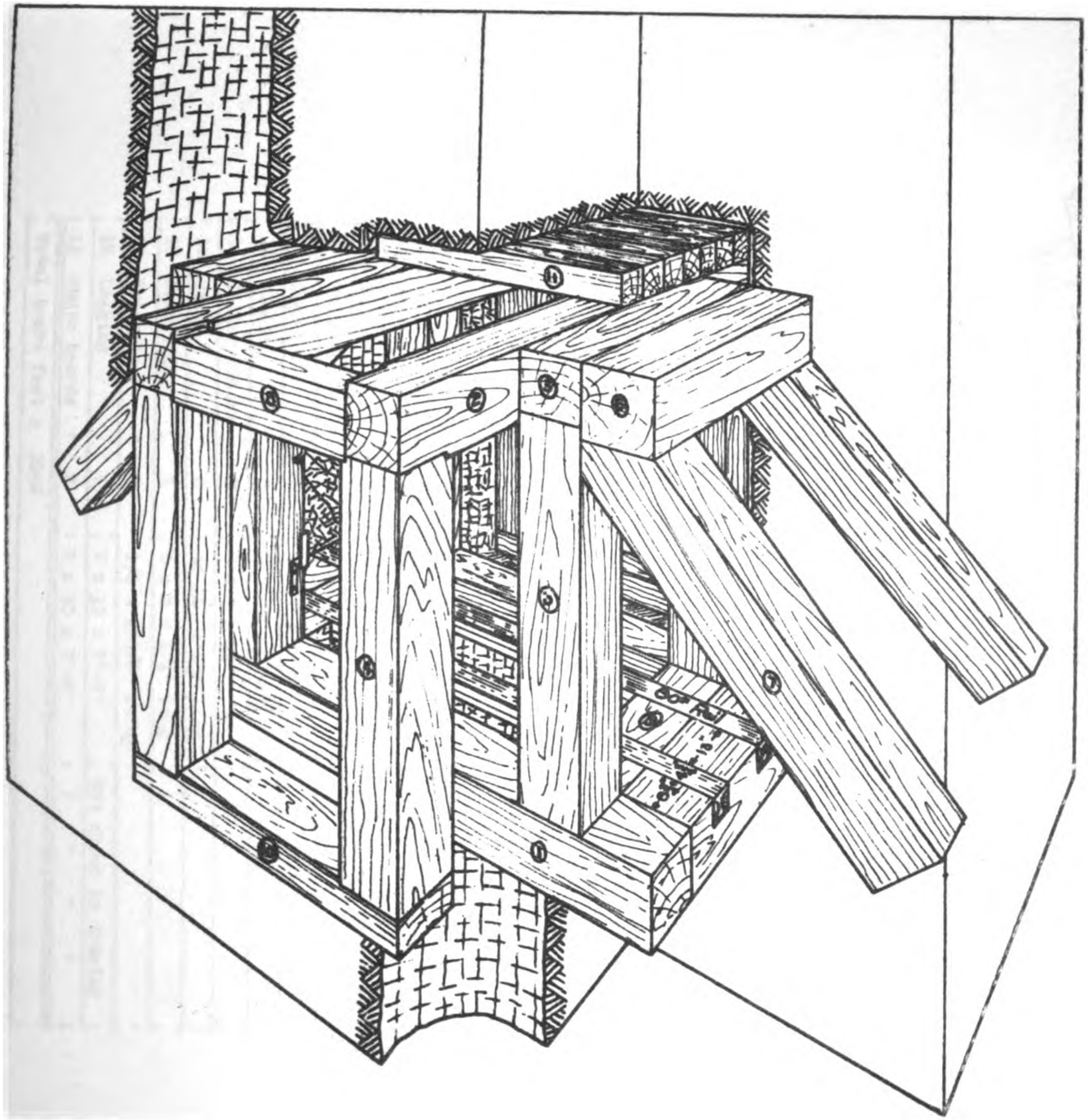
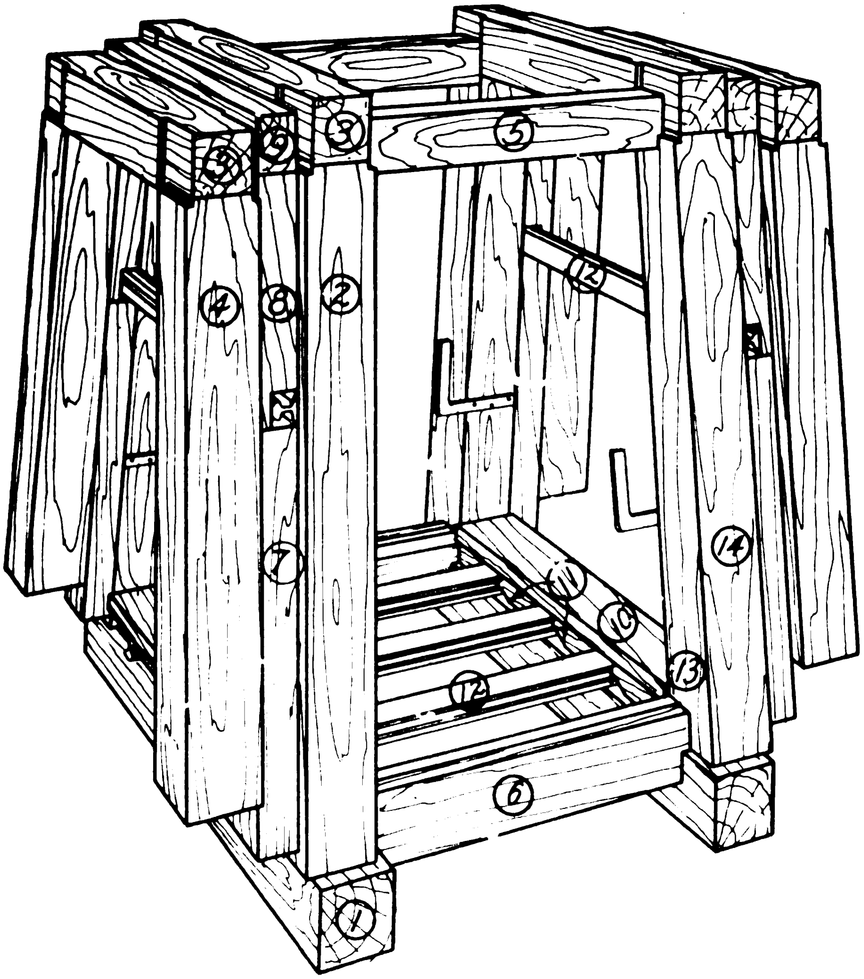


Figure L7.- Grizzly station, Morenci mine



Bill of Material				
№	Pieces	Name	Size	Bd. Ft.
1	2	Stringers	12"x12"x9'0"	216
2	2	Caps	12x12x10'0"	240
3	4	Posts	12x12x5'3"	252
4	4	Posts	12x12x5'3"	252
5	2	Caps	10x12x5'4"	106
6	2	Caps	12x12x5'4"	128
7	4	Posts	12x12x5'6"	240
8	2	Collar braces	8x10x3'2"	42
9	4	Spreaders	12x12x5'4"	160
10	2	Sills	6x12x5'0"	60
11	18	Logging	6x8x5'0"	300
			Total	1996

Figure 18.- Throat set and grizzly for sublevel caving method, Ray mine



Mark	Name	Number Required	Dimensions	Remarks
1	Sills	2	10 x 10	Length to fit ground
2	Posts	4	10 x 10 x 7'-0"	Dap 3 sides 1" at top
3	Caps	4	10 x 10 x 5'-4"	" " " "
4	Posts	4	10 x 10 x 6'-0"	" " " "
5	Collar braces	2	6 x 8 x 4'-0"	
6	Spreaders	2	5 x 10 x 4'-2"	
7	Fillers	4	6 x 8	Length to fit ground
8	Fillers	4	6 x 8 x 2'-4"	
9	Caps	2	6 x 8 x 5'-0"	
10	Planks	2	2 x 12 x 5'-0"	
11	Filler blocks	8	4 x 6 x 14"	
12	Rails	5	40 lb. 6'-0" long	
13	Wear iron	2	3 x 3 x 1/4" long	
14	Brackets	4	1/2 x 1-1/2" strap	
15	Lagging	4	2 x 12 x 5'-4"	Not shown in drawing
16	Chute boards	2	2 x 10 x 5'-4"	" " " "
Total board feet = 1043				

Figure 19.- Grizzly-control set, Inspiration mine

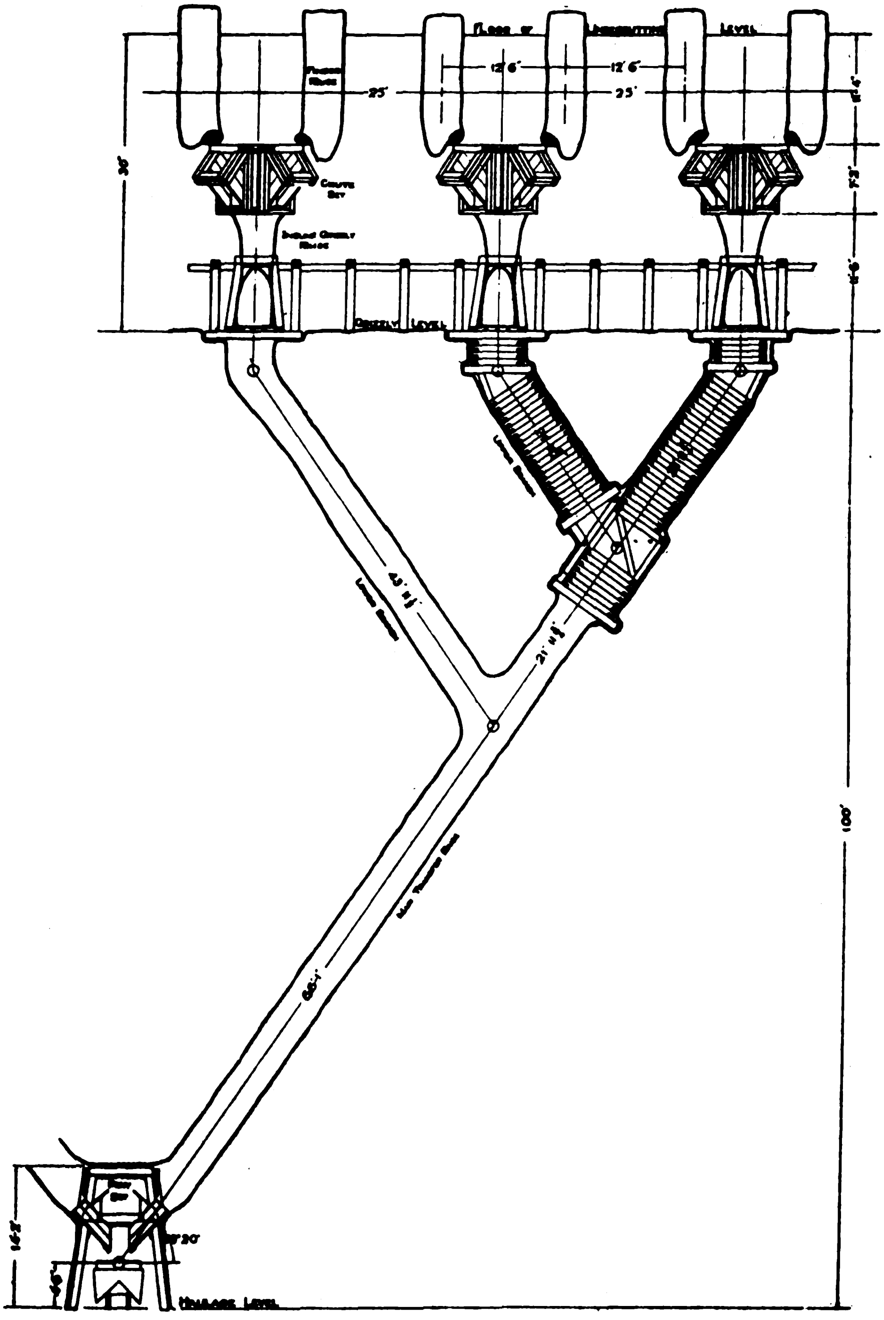


Figure 20.- Vertical projection of ore-transfer raise system, Miami

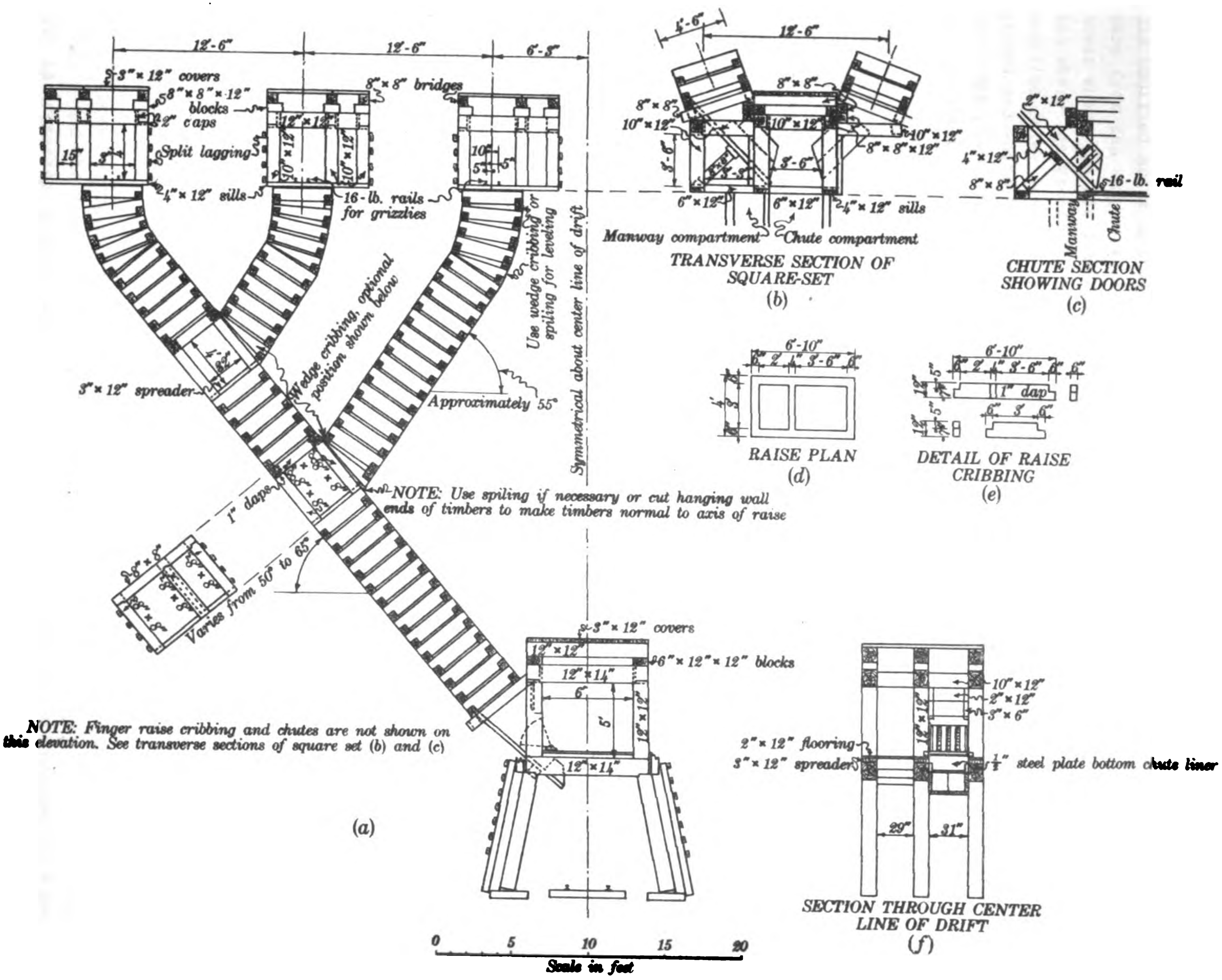


Figure 21. – Model branch raise and timbering details at Ruth mine

Haulage raises should have sufficient storage capacity to make the operations of ore drawing and haulage more or less independent of each other.

At Ray the principal and preferred system of mining is the so-called sublevel method. Under some conditions two variations of this method are used—the motor system and the modified motor system. In the motor system the ore is drawn directly into the haulage cars from the stopes without passing through grizzlies or raises. This system is used in areas where the ore comes down to the haulage level and extends less than 100 feet above it and where the ground is hard and has no tendency to pack after being caved (fig. 14). The modified motor system, with its better spacing of draw points, is used in areas of softer ground and where the height of the ore is less than 150 feet above the haulage level (fig. 15). The so-called sublevel system is used in all types of ground where the thickness above the haulage level is 150 feet or more (fig. 16).

There is no essential difference in the three methods, other than from a development standpoint. The actual mining of the ore is practically the same in all three cases. The sublevel method, in which gathering raises and grizzly levels are used, corresponds more nearly to the methods used at other mines using a caving system. The motor and modified motor variations are used only under special conditions of relatively thin areas of the ore body immediately over the haulage levels.

At Ray and Morenci the haulage drifts are 50 and 56 feet apart, respectively. Single raises from either side of the drift are extended to the grizzly level 40 to 50 feet above.

At Miami, Inspiration, Copper Queen, and Ruth, with wider spacing of haulage drifts, branch raises are used. Because of this feature the mining method has been called the branch raise system. The inclination of the raises varies from 50 to 60°, except at Morenci, where the angle is 75°. Where the ore is soft it will pack in raises that are driven at too steep an angle and hang up in those driven at too low an angle. Experience at Ruth has shown that the best angle for branch raises in such deposits is 55°. The inclinations may vary, within limits, for different heights of raises in the same block in order to bring the ends of the branches at definite intervals on the grizzly drifts.

The tops of the raises at Morenci are widened out immediately below the grizzly level to allow the ore from two draw points to be run in each raise. Figure 17 shows the details of the construction at Morenci. Figure 18 shows the grizzly set at Ray and Figure 19 the one at Inspiration.

Figures 20 and 21 (after F. W. Maclennan - see footnote 13) show gathering or transfer raises at Miami and Ruth. As grizzly drifts are not usually run at Ruth, access to the draw points is through the raises, and manways are maintained in each.

Transfer raises are run 4 by 6 feet in cross section at Morenci. Where the ground will not stand the raises are cribbed. At Ray these raises are 4 by 4 feet in section. If timber is necessary, 6 by 8 inch cribbing is used, giving an inside diameter of 4 by 4 feet.

As the distance between the haulage and grizzly levels increases the distance between the haulage laterals is increased. This results in a decrease in the amount of haulage level drifting and an increase in the footage of transfer raises needed to serve the same undercut area with the same raise interval on the grizzly level. Usually the raising costs less than drifting, which reduces the development cost of the stope. With wider spacing of the haulage drifts and the consequent longer raises there is an increased ore-storage capacity in the raises and less danger in blasting at either end of a raise to the workmen at the other end. Also a block can be developed faster because of less drifting and chute construction, which is slower than raising.

A less distance shortens and simplifies the transfer raise system and provides a more accurate means of measuring and distributing the tonnage drawn. Increasing the number of haulage drifts reduces train congestion.

Grizzly Levels

With the exception of Ruth, a level is run at the elevation of the grizzlies for access to the grizzlies and undercut workings. At Ruth it was found very expensive to keep grizzly levels open and their use was abandoned, except occasionally in favorable ground where the tops of adjoining grizzly sets may be connected with small 3 by 4 foot drifts.

Fringe drifts are usually run along one side of a panel or entirely around the blocks, and the grizzly drifts turned off at right angles. One fringe drift may serve two panels. Tracks are usually maintained in the fringe drifts.

The grizzly level is the center of the operations after production starts, so that any interruption due to caving of drifts and repair work is very costly. It has, moreover, been found that the workings on this level are the most liable to damage from weight developed in undercutting and caving. To prevent this, very heavy timber is used and the size of workings is kept small; the latter is quite effective. Frequently a small untimbered grizzly drift will remain open when a larger drift could not be maintained with even heavy timbering. It is usually considered essential to eliminate "overbreak" at the grizzlies. At Morenci one round of winze is blasted for the top of the transfer raise, to avoid the large opening generally made when connecting a raise to the bottom of a drift and to avoid the results of faulty direction.

Grizzlies

The grizzly-bar spacings limiting the size of bowlders that pass through them should be as wide as the subsequent operations of haulage and hoisting will permit. When the ore to be handled is naturally coarse the spacings should be planned to handle large bowlders conveniently. Chute-tapping labor is thus reduced and the speed of drawing increased.

Grizzlies are placed in sets, as shown by Figures 17, 18, and 19. The sets are usually 4 by 4 feet or 4 by 6 feet in plan. The bottom of the set is occupied by the grizzly. The grizzly bars are usually rails set in heavy timbers. The following tabulation shows the weight of rails used and the width of the opening between the rails at a number of mines.

Weight and distance between grizzly rails

Mine	Weight, pounds per yard	Width of opening between rails, inches
Morenci	50	16
Copper Queen	75	16
Ray	80	13
Inspiration	40	10
Miami	45	10
Old Dominion	24	10
Ruth	16	8

The spacing of the grizzly bars is in the main a measure of how the ore breaks up in caving at the different mines. At Morenci the average ore is relatively hard and breaks coarsely; at Ruth the ore is soft and on caving usually breaks down in fragments the size of sand and gravel and tends to flow.

Draw Points

The place where the ore is drawn after being undercut and caved is called a draw point. These points are subjected to great pressure. As far as possible they should be protected from weight in order to permit continuous drawing and to avoid repairs which would interrupt drawing and increase the cost of maintenance.

Weight is resisted most efficiently by leaving as much unbroken ground as possible about the grizzly drifts to act as pillars. This can be accomplished by keeping the size of the grizzly drifts small and by cutting down the number of finger raises. Wide spacing of the fingers is limited by the requirements of good drawing, which prescribe spacing them as closely as possible.

The proper spacing of draw points for each ore body usually has to be worked out by trial to obtain the proper balance between close control of drawing and the cost of development work and maintenance of workings. Draw points in soft ore that tends to pack or chimney are usually placed on 12 1/2-foot centers. At Inspiration in the original ore body the draw points are spaced 12 1/2 feet apart one way and 10 5/8 feet the other.

The Inspiration Consolidated Copper Co. has done considerable experimental work in determining the best spacing and arrangement of draw points. Spacing of draw points as close as 8 1/3 feet apart was tried in the original ore body but was found not practical because insufficient support was left. Experiments with the arrangement of draw points where close spacing is required indicated that one raise out of each of the four sides of the draw set gave the best results. In the harder ground of the Live Oak division wider spacing than that used in the original ore body has been found satisfactory.

Before the ore is undercut and caved raises are run from the draw points to the undercutting level and usually belled out at the top. Two general kinds of draw sets are used: A chute set in a raise above the grizzly with finger raises from the four sides, and sets on either side of the grizzly drift or raise at the grizzly from which single raises have been run to the undercut level. The flow of ore is controlled usually by means of simple chute gates consisting of plank resting in iron brackets (fig. 19) or wooden cleats nailed on the inside of a timber set.

The ore from eight points is drawn through each grizzly with the chute-set method and from only two with the other. The chute-set system has the advantage in that closer spacing of draw points is possible, which allows closer control of the ore. Its principal disadvantage is that two men are required to draw the ore—one in the chute and one at the grizzly. Another disadvantage is that the man in the draw set works in a constricted space, where he is more likely to get caught by a run of ore. Supervision of the drawing is more difficult in the draw sets than at the grizzly level. The draw sets are expensive to maintain.

Figure 22 shows the chute set used at the Inspiration mine. The raises from the drift to the chute set are 18 feet long and are driven on an inclination of 40 to 42° at right angles to the drift. The set is installed on an angle of 60° from the line of the drift. Finger raises are run from the four sides of the set which connect with the undercutting level and extend to a vertical height of 19-1/2 feet above the bottom of the chute set. The ends of the opposite finger raises from the chute sets are 18 feet apart center-to-center. Draw points from alternate rows are staggered (fig. 23). The area drawn from each point is in the shape of a pentagon.

At Miami, the system is quite similar and was adopted for mining an ore body of similar physical characteristics as the one at Inspiration. The chute sets are placed at 45° to the line of the drift, which allows even spacing; the draw points are in line (fig. 24 - after F. W. Maclennan, footnote 13). The area tributary to each draw point is a square. This is also true at Ruth.

At Ruth the chute sets are at the ends of raises from the haulage level (fig. 21). Two finger raises are driven from each chute set at right angles to the plane of the branch raises. At the undercutting level the tops of the finger raises are on 12-1/2 foot centers (section BB, fig. 25). Careful surveying is necessary to bring the tops of the finger raises at the properly-spaced intervals.

At the Andes Copper mine chute sets normal to the grizzly levels are used. Two of the finger raises are run at an angle to the set to allow for even spacing of the draw points (fig. 26).[15]

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[15] - Greninger, I. L., Mine Development and Underground Construction of Andes Copper Mining Co. at Potrerillos, Chile: Trans. Am. Inst. Min. and Met. Eng., 1929, pp. 144.

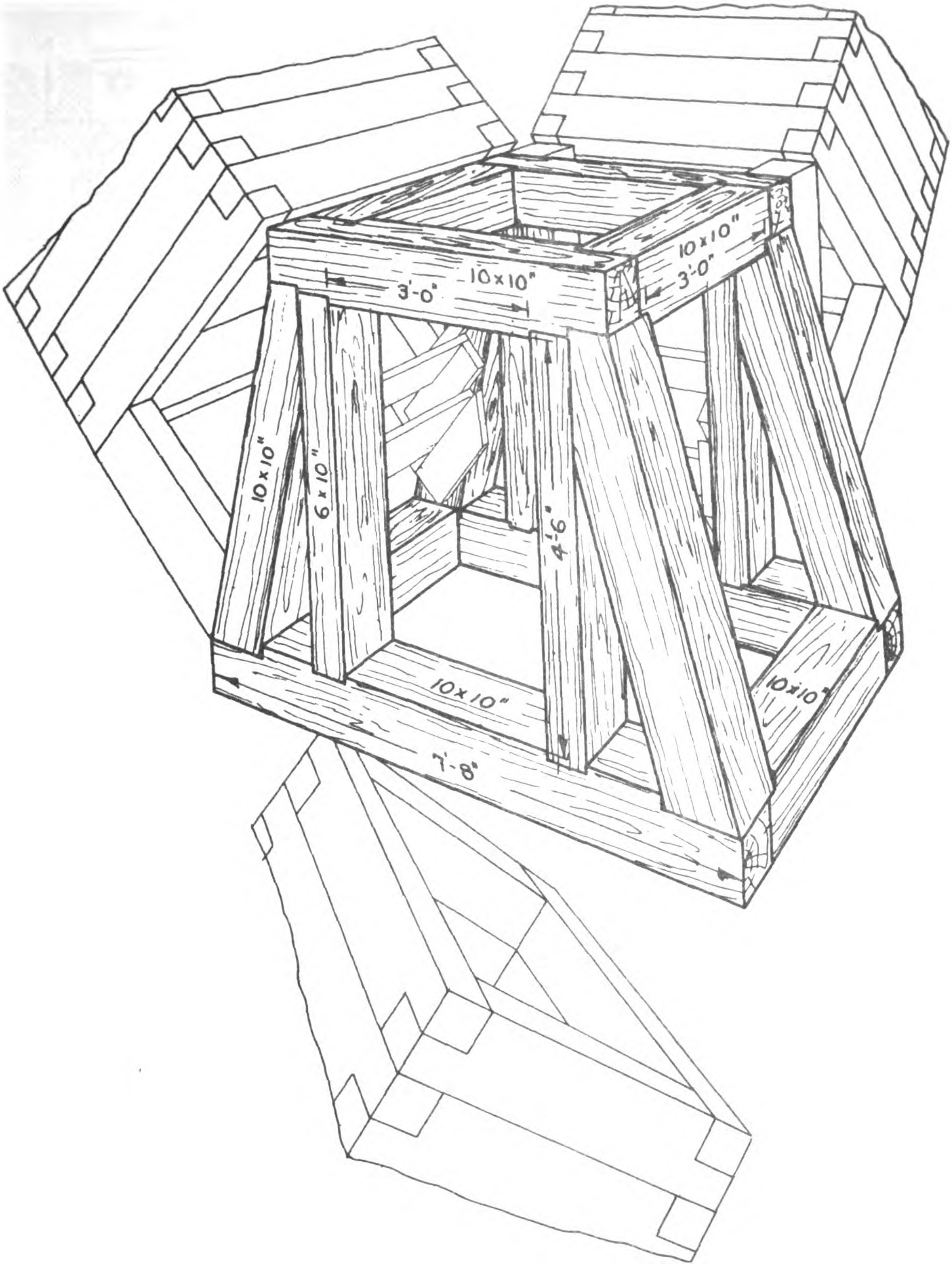


Figure 22.- Chute set for control of drawing, Inspiration mine

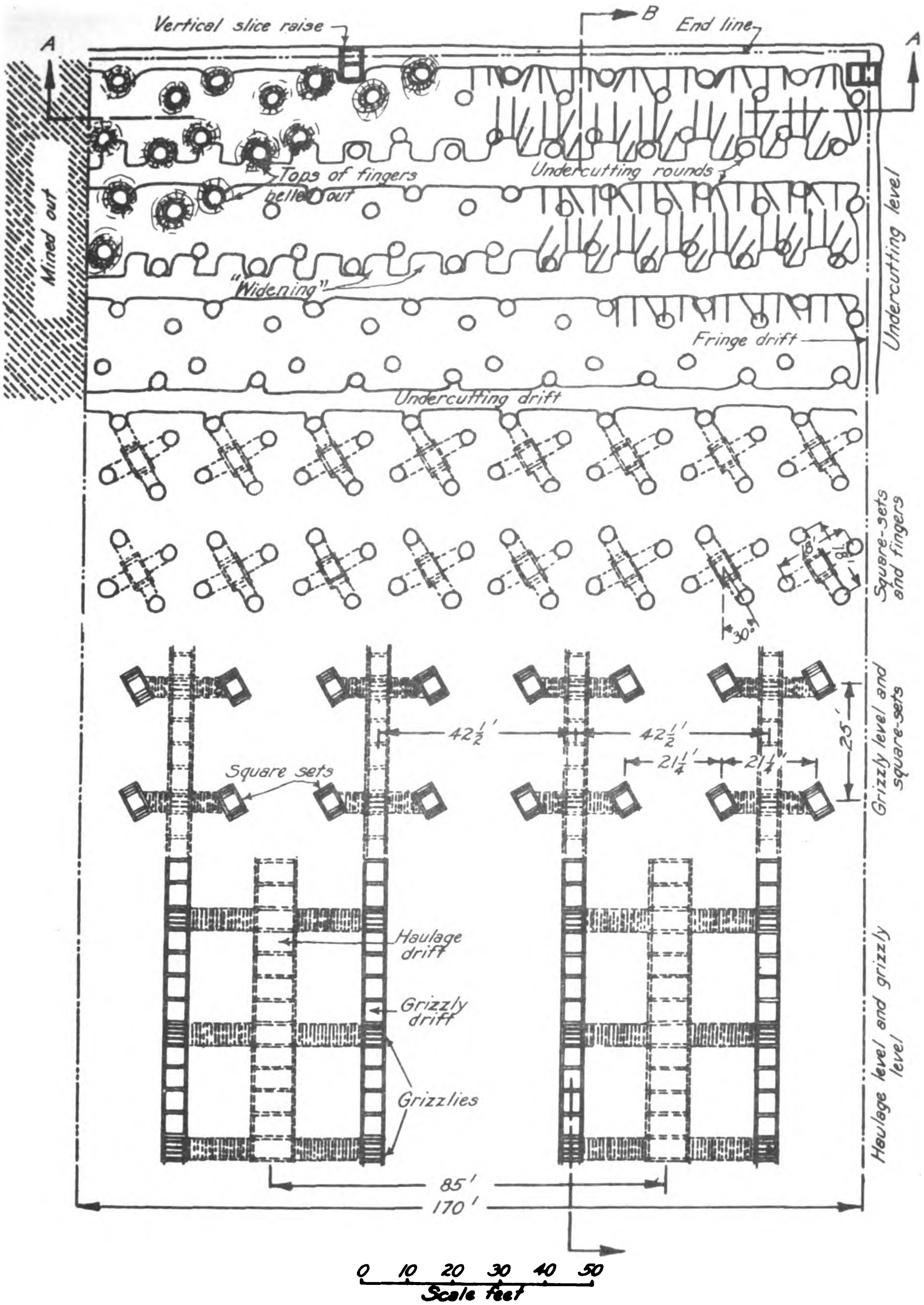


Figure 23.- Plan of caving method using chute sets , at Inspiration mine

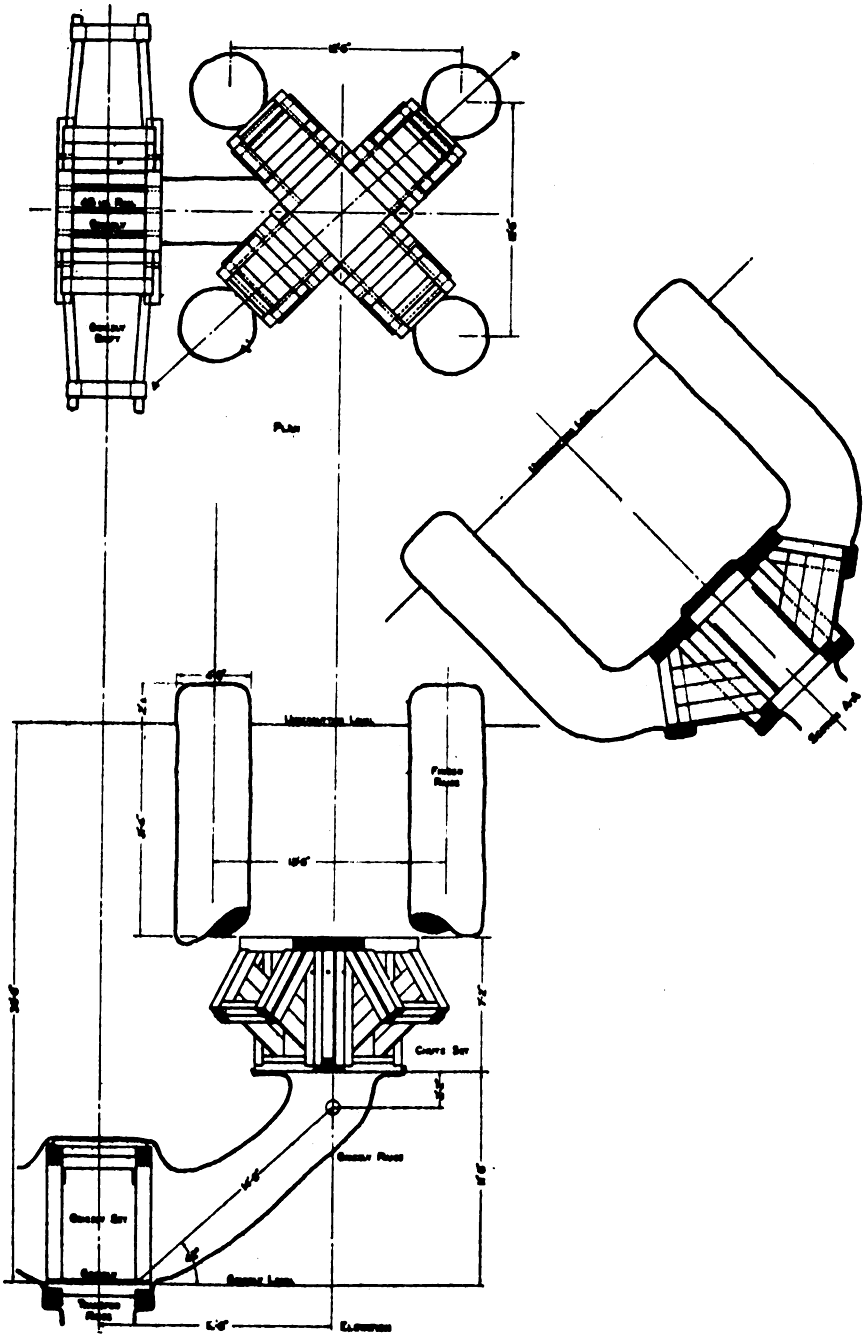
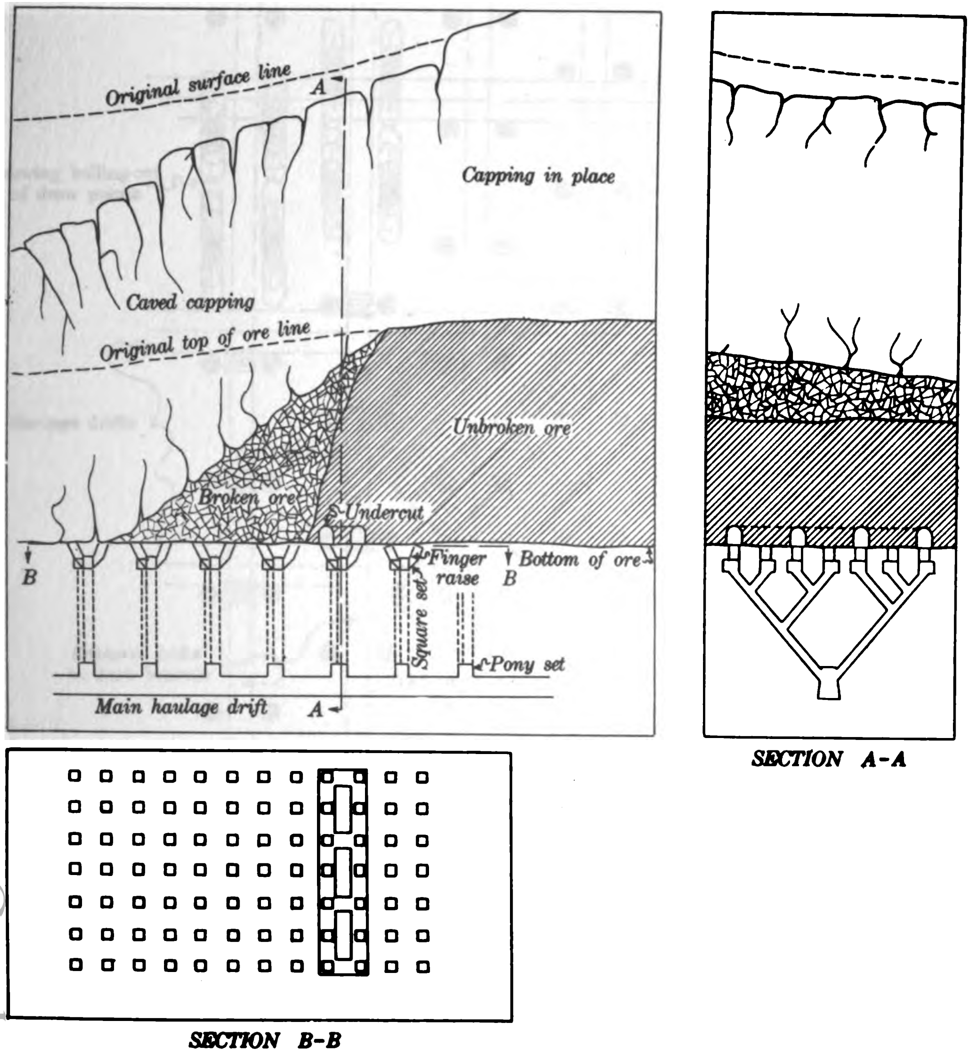


Figure 24.- Location and orientation of chute set at Miami



SECTION A-A

SECTION B-B

Figure 25. – Caving system used at Ruth mine

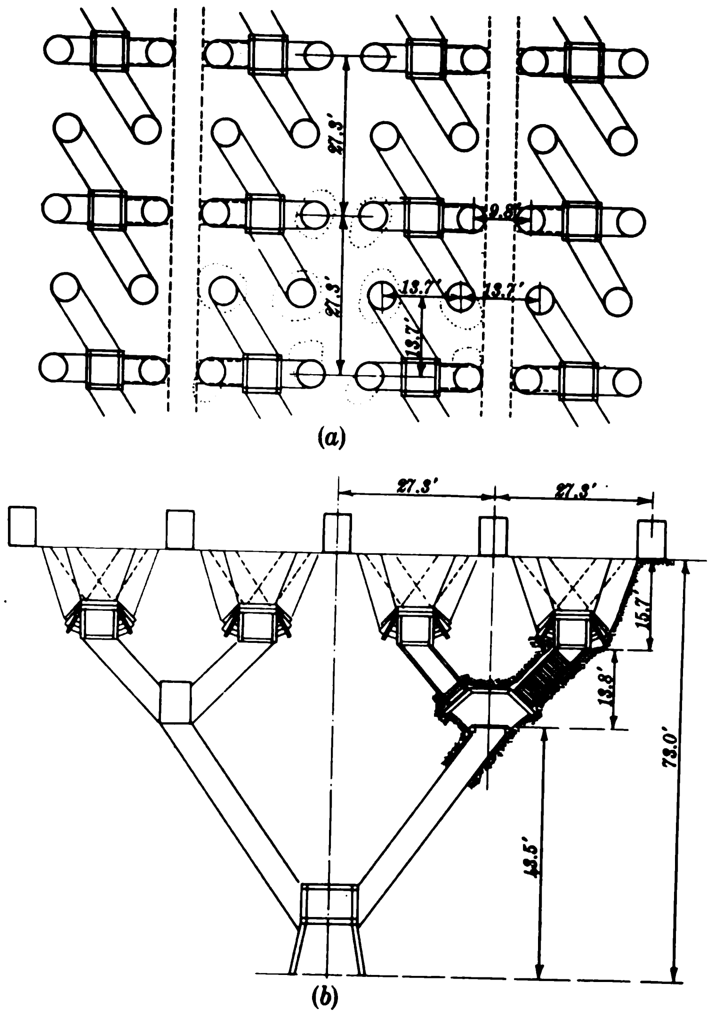


Figure 26. – (a) Plan of undercutting level showing top of branch raises.
(b) Transverse section on main and branch raise system, Andes Copper Co.

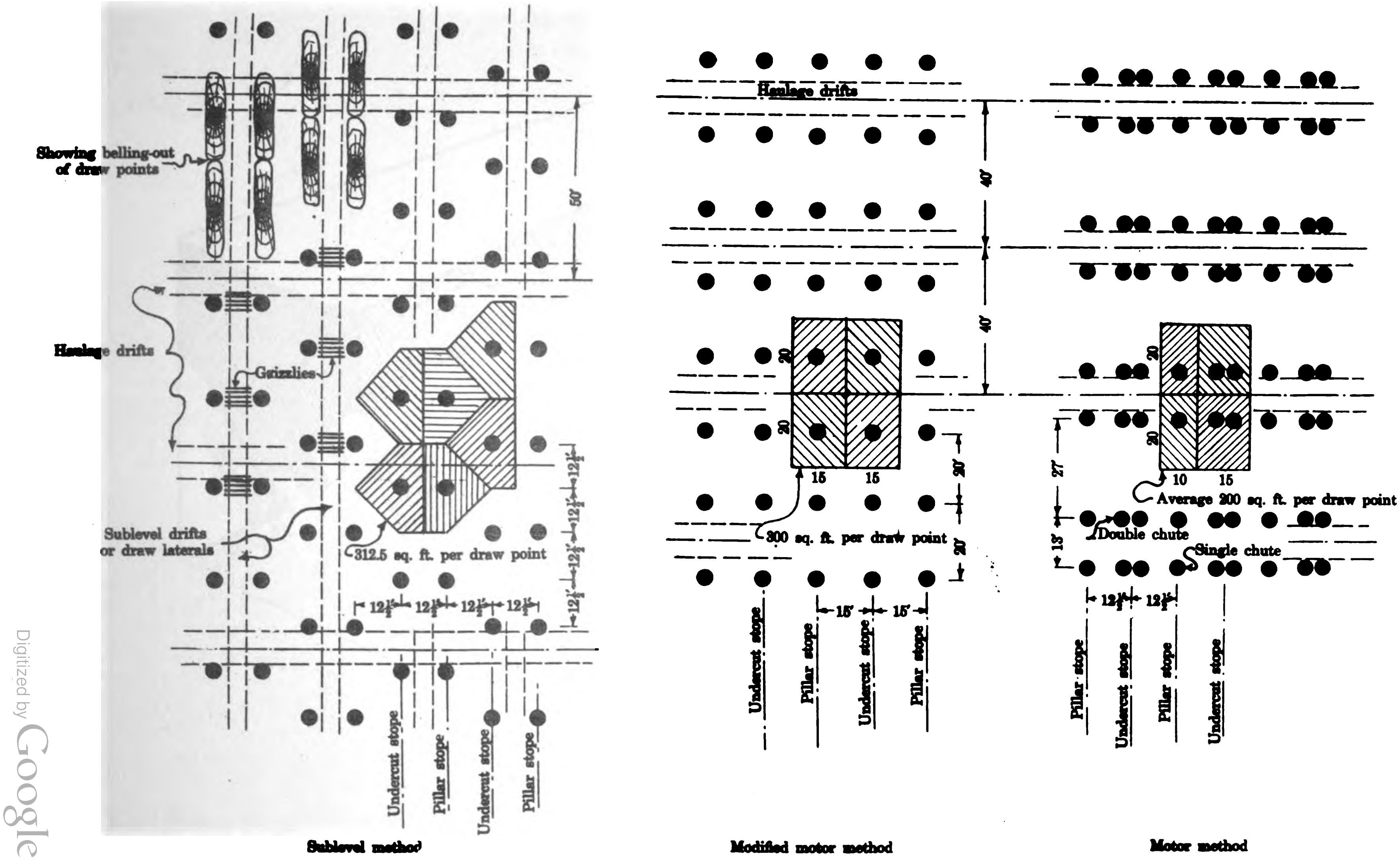


Figure 27.- Diagram of draw-point spacings at Ray

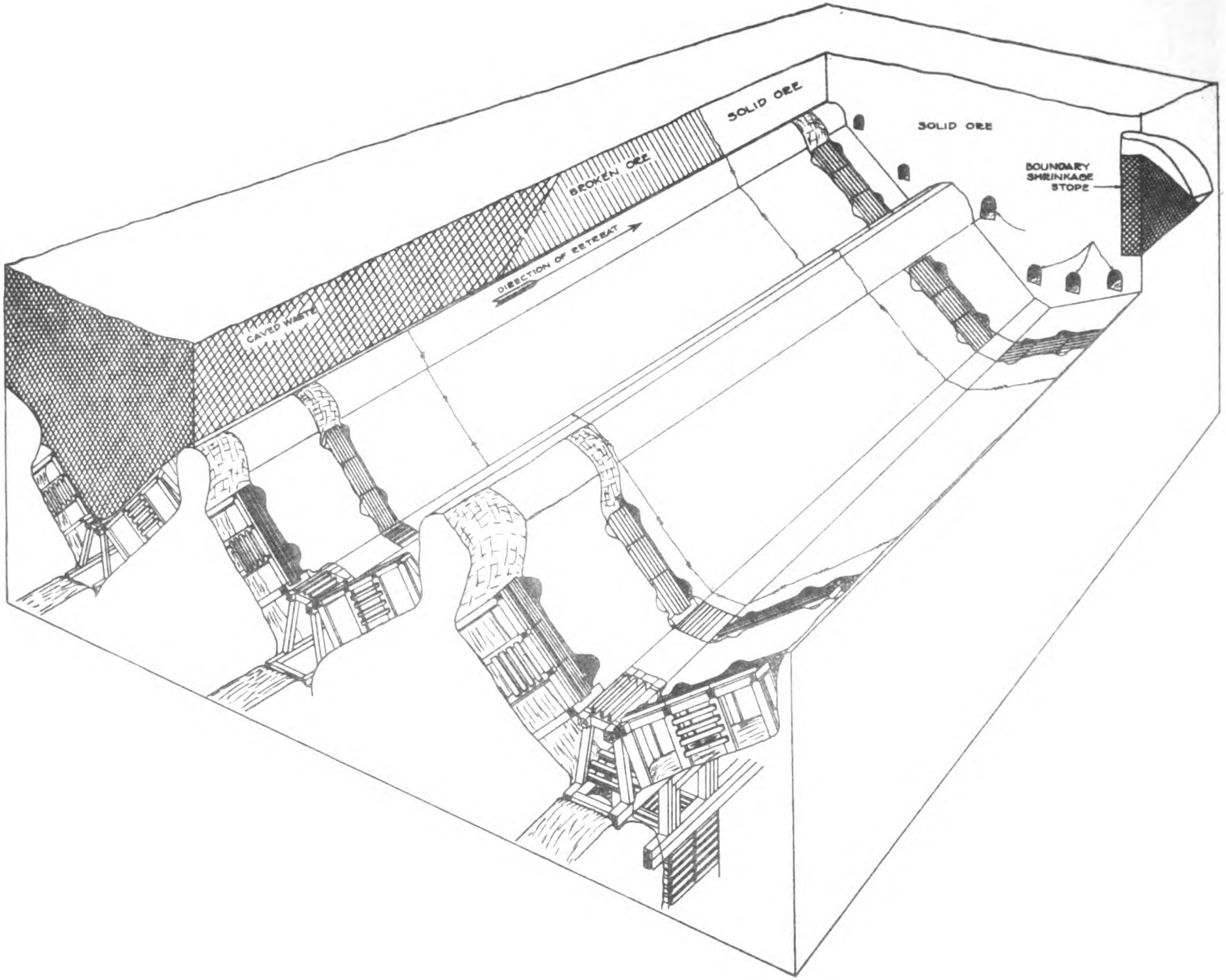


Figure 28.- Perspective of Morenci slide method of mining, 2300-level ore body, Old Dominion mine

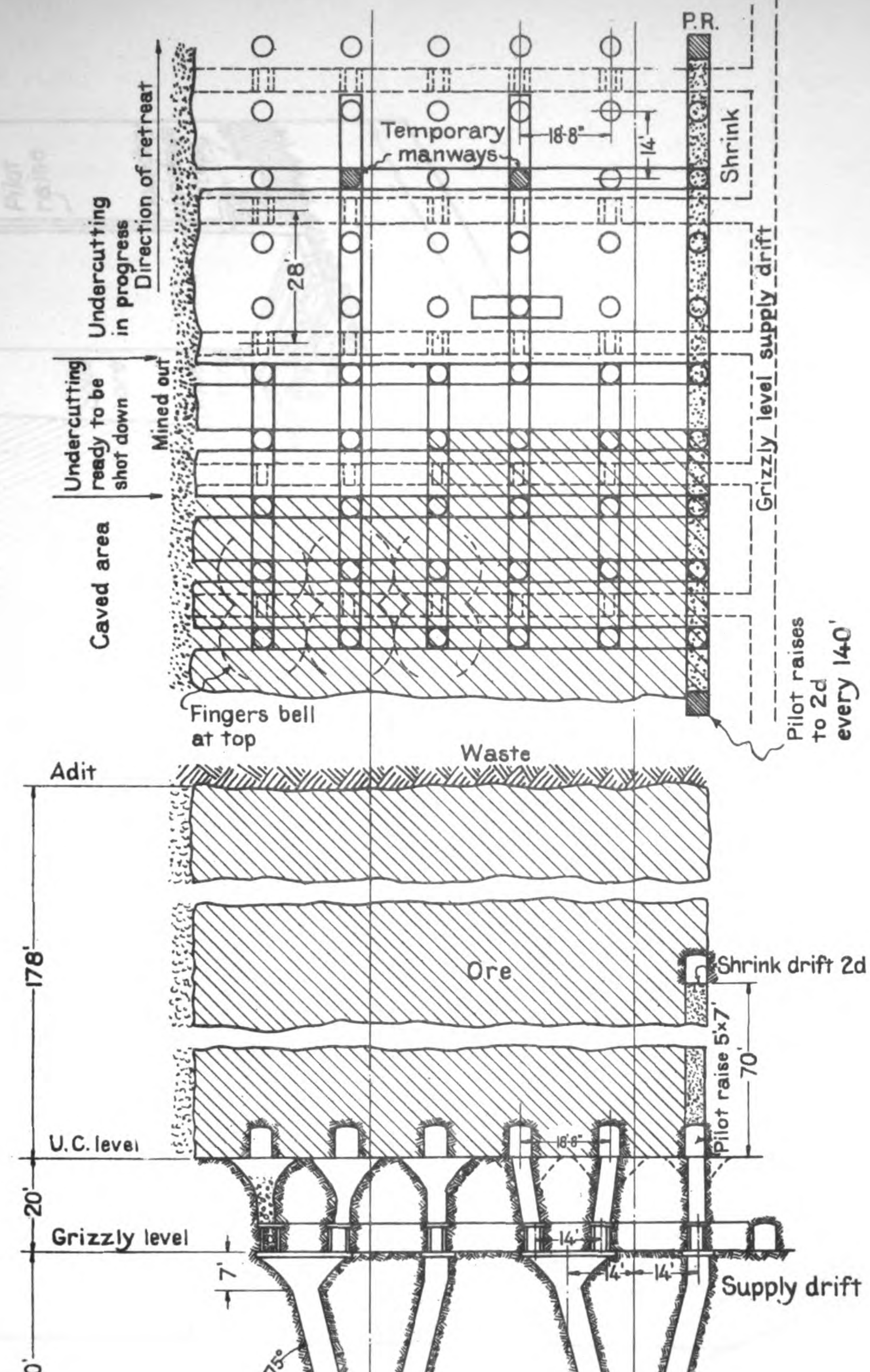


Figure 29.- Standard Morenci block-caving method

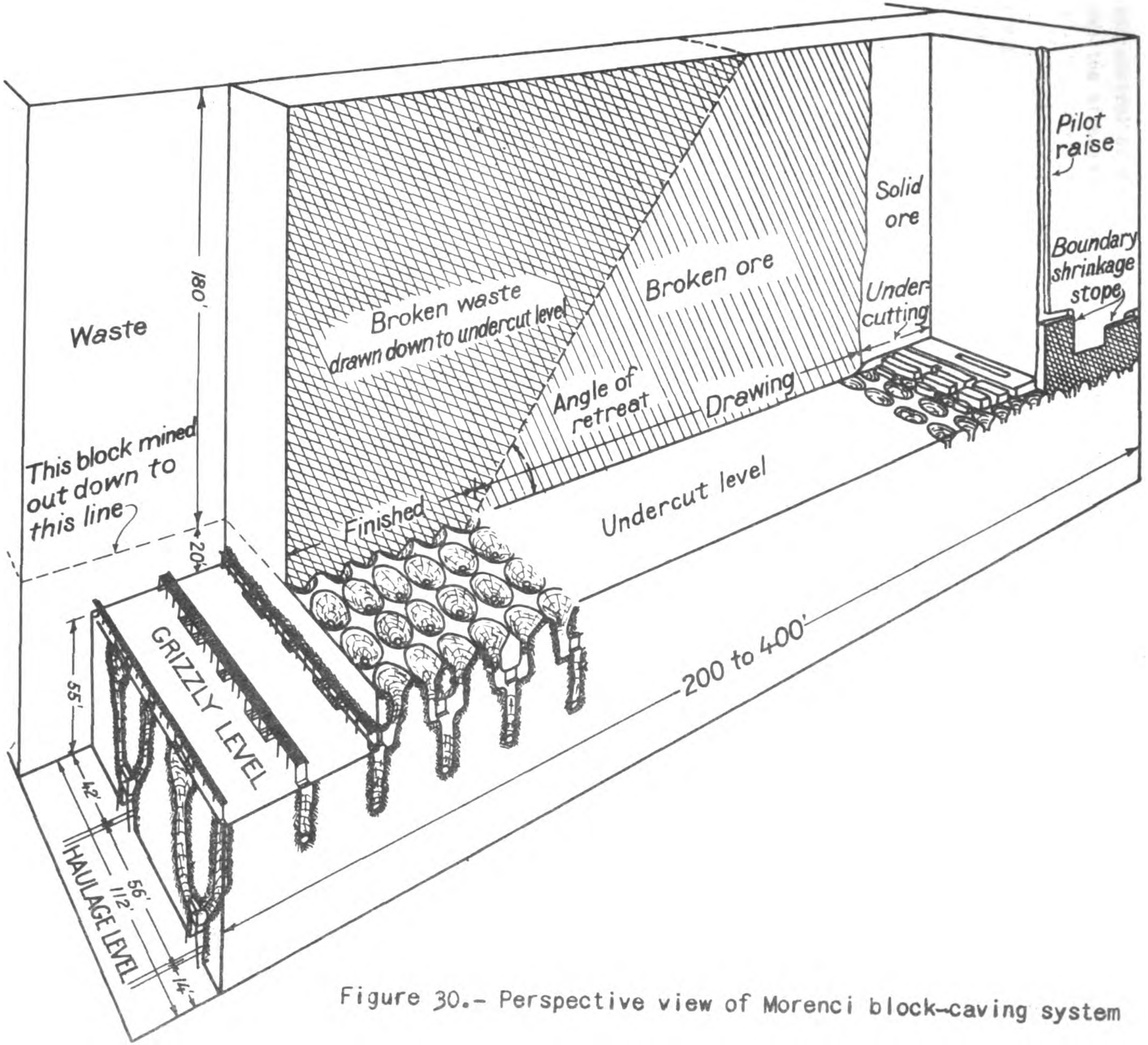


Figure 30.- Perspective view of Morenci block-caving system

As at Ray, Morenci, Copper Queen, and the Live Oak division of Inspiration the ore is harder and stronger than at Miami and Ruth, or in the original Inspiration ore body, such close control of drawing is not needed, and draw points at the grizzlies are used. Figure 27 shows the arrangement of draw points at the Ray mine.

At the Old Dominion the Morenci slide method is used. Incline raises or slides correspond to the grizzly drifts at the other mines; the ore is drawn from the sides of the incline raise (fig. 28).

METHODS OF CONFINING STOPES TO PREDETERMINED LINES

Before the ore is mined in blocks it is necessary to devise a method to cut it free from adjoining ground and guide the lines of shearing to the top of the ore in the desired planes. The method used for cutting off the blocks or panels depends upon the character of the ore and the height of the lift. In some places the block must be almost entirely cut off on all sides by means of narrow shrinkage stopes. At others the boundaries need only be weakened. The main object, as stated before, is to form vertical planes of weakness on the sides of the blocks to prevent the backs of the stopes from forming self-sustaining arches which would obstruct the caving; at the same time, it is necessary in some cases to avoid weakening the sides or ends to such an extent that the back would drop out in huge blocks or possibly in a single large block. With low lifts shrinkage stopes around a block might be the most satisfactory method of isolating it. However, with higher lifts in the same ore such a method might let the block come down as a single mass. A complete separation exists on sides adjoining finished stopes and of course no further cut-off workings are needed.

Samples are taken in boundary cut-off workings to allow an estimate of the grade of the ore in the panel or block and to aid drawing operations. Knowledge of the physical character of the ground in these cut-off workings is also valuable during drawing operations. The extent of such knowledge to be gained of course depends upon the extensiveness of these workings.

At Ray both end and side cut-off shrinkage stopes around new blocks are carried up to the capping or to a previously-mined area above. In blocks with high lifts the cut-off stopes are run in two sections, the upper half from a sub-level (fig. 16).

At the Old Dominion shrinkage stopes are run for cutting off the ore from the surrounding waste (fig. 7).

At Morenci, where the character of the ore approaches that at Ray, shrinkage cut-off workings are first started by driving raises 140 feet apart to connect with an old intermediate mine level 70 feet above the undercut level. In weak ground shrinkage stopes are then carried up 35 feet. At this elevation it has generally been found possible to leave intact sections 50 to 60 feet in length, situated midway between the pilot raises, and to continue shrinking upward only in sections 40 to 50 feet on each side of the pilot raises. This results in a saw-tooth arrangement of broken and unbroken ground (fig. 29 and 30). These shrinkage sections are carried up to the elevation of the intermediate level, and if necessary to weaken the boundary still further they may be driven higher. Enough ground must be cut through in the boundary cut-off workings to permit the blocks to begin caving without delay after undercutting, but not enough to cause the whole block to collapse at once. The continuous stope taken for the lower 35 feet of the block boundary assists the caving. Leaving part of the ground in place on the boundary above prevents caving too fast and in large masses.

Morenci ore varies so much that no constant percentage of the area on the boundaries can be used to decide how much shrinkage stoping is to be done. That must be left to the judgment of the operator. It is better to cut out too much, even at the risk of a sudden collapse, than too little and have the block hang up indefinitely. Moreover, sections of unbroken ground on the stope boundaries should not be so great that the line of shearing may diverge from the desired plane.

At Inspiration raises are run at the corners of each block. Other raises are put up on the boundary at about 110 feet from the corners. Drifts 30 to 40 feet apart vertically are run around the block. These boundaries may be further weakened by blasting a stope round in the back of the drifts, depending upon the nature of the ground (fig. 31).

At Miami, the boundaries of the block are weakened by driving boundary caving drifts completely around the stopes at suitable vertical intervals (fig. 32), and putting up raises at all four corners of the stope to avoid lateral arching at these points. These raises also serve as an outlet for broken ore and for ventilation during development. The boundary caving drifts are 7 1/2 feet high are were originally located at 30-foot vertical intervals. This had the effect of weakening the plane of the stope boundary 25 per cent; when desired in hard ground the plane can further be weakened to 50 per cent inexpensively by drilling and shooting in the back of the drifts. The present practice, as a matter of economy, is to put these drifts at 45-foot intervals and to shoot the backs in, thus weakening the boundary plane 33 1/3 per cent.

Geological maps of each boundary caving level are prepared, which show the principal slips, inclusions of capping, oxidized ore, etc. These maps are helpful in the control of ore-drawing operations. For example, reference to these maps may show that the capping which appeared early at some of the draw chutes was coming from an inclusion low down in the ore body and not from the overlying capping and that ore drawing from these chutes should be continued, whereas without this information they would be sealed.

The boundary caving drifts afford a means of access to the corners of the original stopes, from which the actual fracturing and caving down of the back of the stopes may be observed at successive elevations as it progresses upward.

At the Copper Queen the blocks are cut off by a series of drifts and raises which leave unbroken pillars 20 feet long and 15 feet high along the boundary. In hard ground rounds can be blasted in the back of the drifts to weaken the boundary further or the pillars can be blasted out, making an open shrinkage stope (fig. 33).

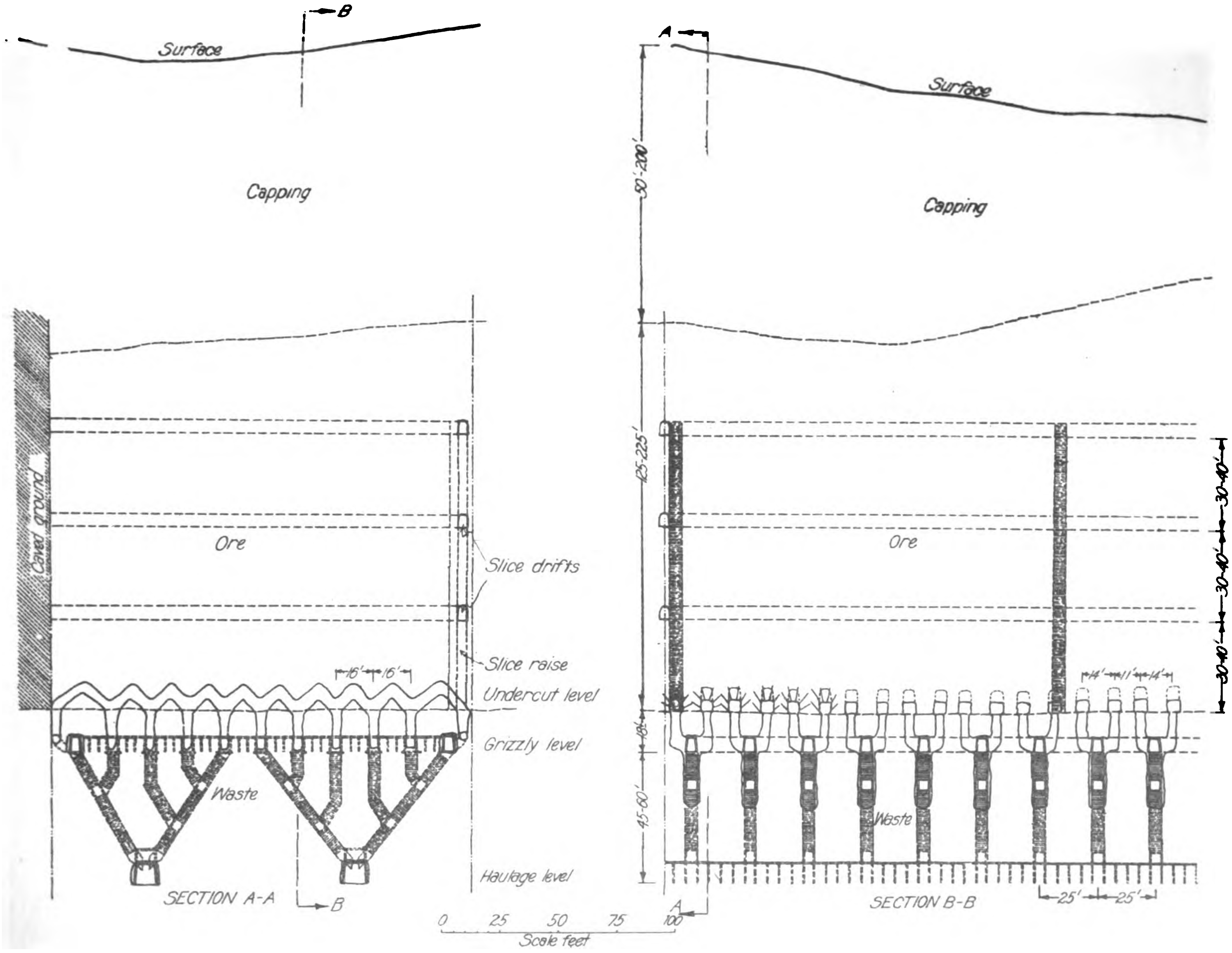
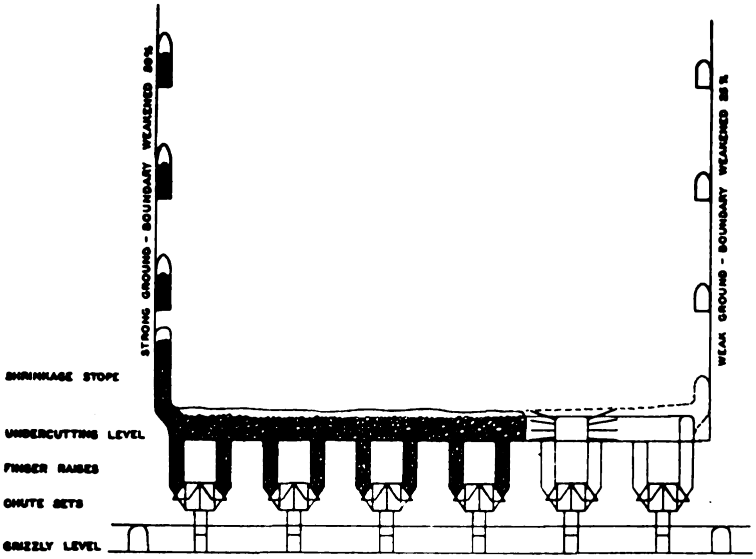
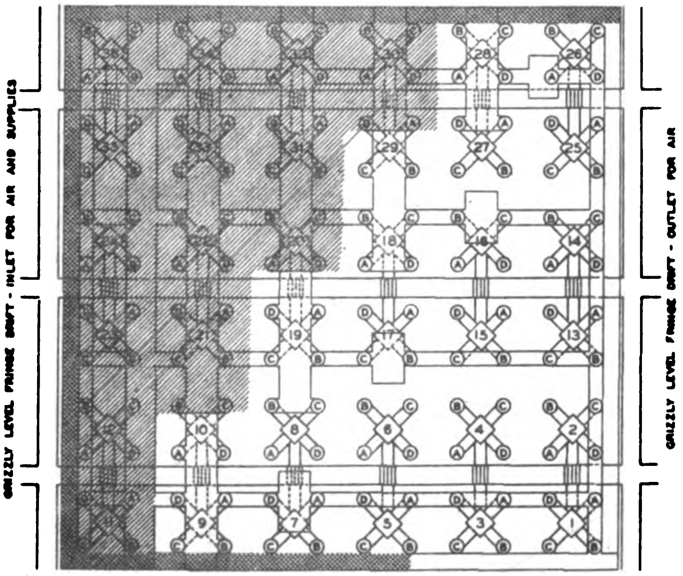


Figure 31.- Sections showing grizzly-control caving method, Inspiration mine


STRONG GROUND - BOUNDARY WEAKENED 80%
WEAK GROUND - BOUNDARY WEAKENED 25%
SHRINKAGE STOPE
UNDERCUTTING LEVEL
FINGER RAISES
CHUTE SETS
GRIZZLY LEVEL

COMPOSITE SECTION


GRIZZLY LEVEL FRINGE DRIFT - INLET FOR AIR AND SUPPLIES
GRIZZLY LEVEL FRINGE DRIFT - OUTLET FOR AIR
26
28
25
27
29
14
16
18
13
15
17
19
2
4
6
8
10
1
3
5
7
9

COMPOSITE PLAN
OF
UNDERCUTTING AND GRIZZLY LEVELS

▨ MAIN UNDERCUT
■ BORDER SHRINKAGE STOPE.

Figure 32.- Method of undercutting at Miami mine

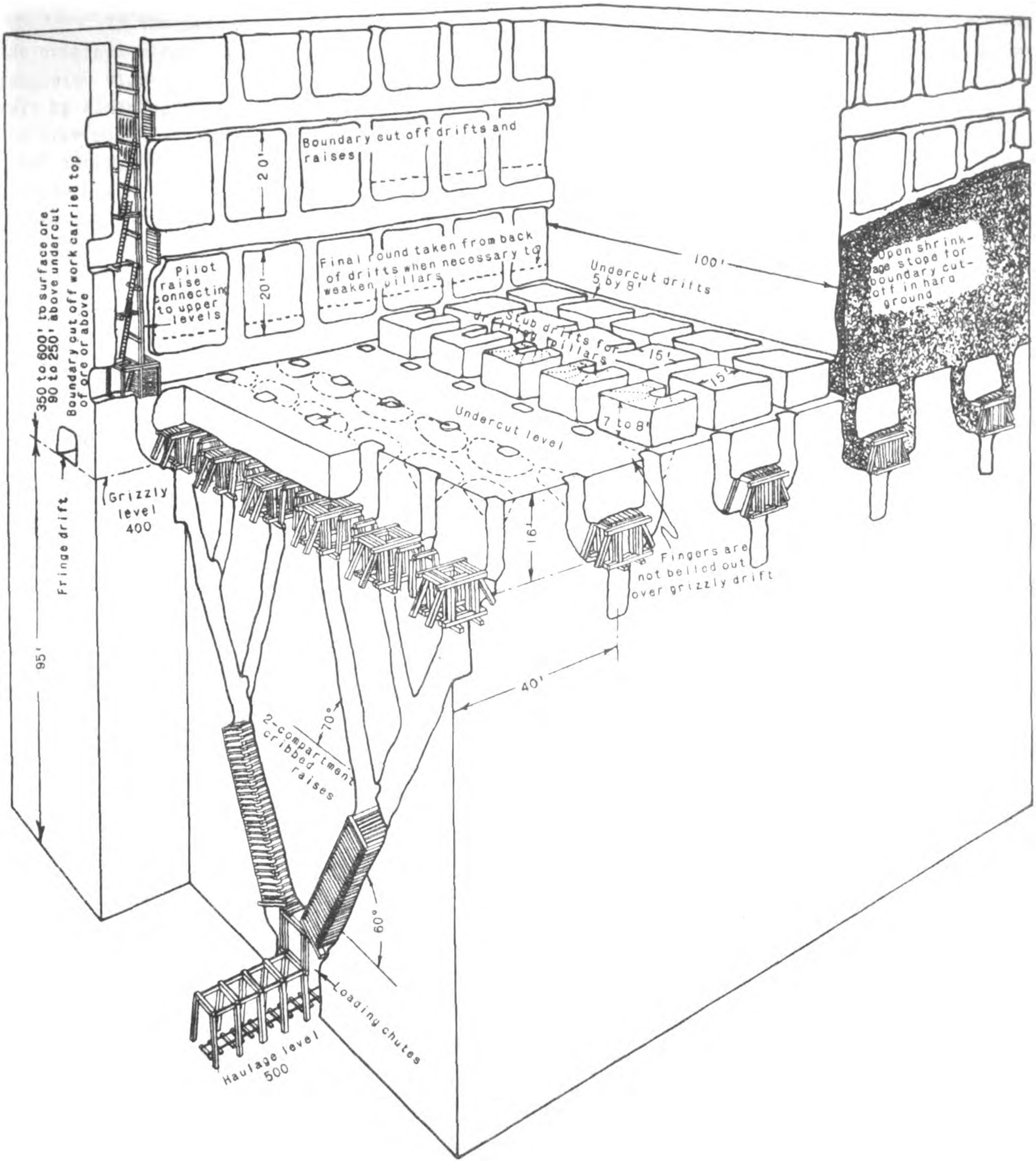

350 to 600' to surface ore
90 to 250' above undercut
Boundary cut off work carried top of ore or above
Boundary cut off drifts and raises
20'
Pilot raise connecting to upper levels
20'
Final round taken from back of drifts when necessary to weaken pillars
100'
Undercut drifts 5 by 8'
Open shrinkage stope for boundary cut-off in hard ground
Stub drifts for drilling pillars
15'
15'
7 to 8'
Undercut level
Fringe drift
Grizzly level 400
16'
Fingers are not belled out over grizzly drift
95'
40'
70°
2-compartment cribbed raises
60°
Loading chutes
Haulage level 500

Figure 33.- Perspective of undercut block-caving method at Copper Queen mine

Partitions Between Stopes

In the present method of mining at Miami 15-foot partitions of ore are left between the original stopes and the pillar stopes to prevent dilution from the settled capping in the completed stopes. Some of the later partitions have been 7 1/2 feet thick. The partition is left by widening the distance between the border draw point of the pillar stope and the border draw point of the original stope adjoining. The partitions amount to 11 per cent of the total tonnage of the ore body served by the 720-foot haulage level at Miami and are not included in tonnage estimates. Extraction records, however, indicate that some of this ore is recovered. Experience gained in using a panel system and retreating across the entire ore body indicated the advisability of adopting this practice. Partitions are not used at the other mines discussed in this paper.

For the purpose of preventing side dilution a number of mines follow the practice of drawing the ore adjoining a completed stope last.

METHODS OF UNDERCUTTING

In general the stopes are undercut by running either a checkerboard system of drifts and crosscuts and blasting the pillars, or by belling out the finger raises to intersect, or by a combination of both methods. A system that will insure a complete undercut should be used in hard ore; otherwise the back may hang up and not cave. Usually after the pillars are shot it is impossible to get back in the undercutting level to redrill or reblast unbroken stubs of pillars or shoot missed holes. In weak ore this is not so important, as any pillars not completely broken by blasting are usually broken by the weight of the ore. In harder ground, however, unbroken pillars may prevent the stope from caving. Getting stopes to cave that have failed to come down after the undercutting has been finished is an expensive operation, and at times such blocks have been temporarily abandoned and adjoining areas mined first. Where the ore is mined in panels, transverse shrinkage stopes may be run across the panels to induce caving.

A small area of ore under a stope that remains unbroken may act as a pillar by which excessive pressure can be transmitted from the broken ore to the grizzly level. The layer of the ore broken during undercutting must be thick enough to allow caving to begin.

At Morenci, in beginning the undercut, which follows just behind the boundary shrinkage workings, one or two drifts on the undercut level are run lengthwise with the panel across several grizzly lines for convenient access to the undercutting drifts. Undercutting drifts 4 by 6 feet in section, locally termed "dog holes," connect the top of the belled-out finger raises both laterally and longitudinally, thus forming a gridiron of drifts and leaving rectangular pillars between about 10 by 14 feet in plan (fig. 29 and 30). Certain fingers are driven in advance for handling supplies for the undercut section and for ladderways from the grizzly level, thus insuring a safe horizontal retreat for the men on the undercut level.

The checkerboard is developed, beginning at the area already caved, by connecting one or two rows of fingers across the panel and proceeding in the direction of the stope retreat. After the drifts are completed the pillars are drilled with hand-held plugger drills.

The backs of all "dog holes" are drilled to a depth of 6 feet in fan-shaped rings of stoper holes at 4-foot intervals. These holes are fired simultaneously with the pillar shots. The extra cut assists in starting the caving action and with the open finger raises provides room for the expansion of the broken ore.

The pillars are usually blasted down two or three at a time, beginning in the heaviest ground and progressing across the stope. Where conditions permit, this blasting is delayed until the pillars begin to show strain. When a pillar begins to take weight before blasting immediate caving of the back is assured. Blasting is done with No. 8 instantaneous electric detonators, which have eliminated difficulties arising from cut-out holes as well as the hazards from loose powder in the ore drawn from the stope.

The advantages of the checkerboard system of undercutting are: (1) The safety of the workmen who work in low drifts. There is no danger of drilling into a missed hole, for the second half of a pillar is drilled before the first half is shot. (2) The ventilation is good. Fresh air is drawn from the grizzly level through open fingers and is carried up through broken ground in the boundary cut-off workings to a primary exhaust air course on the intermediate supply level. (3) The method is elastic since the pillars showing the most weight may be shot out first, regardless of their position in a line. (4) Immediate inspection of results is made possible by shooting only a small number of pillars at one time.

In opening a new panel 112 feet wide it is necessary to undercut through a length ranging from 84 to 196 feet to break up arches or cantilevers and to start caving freely, even though the new panel may lie against the caved waste filling in a completed section.

At Inspiration in the original ore body undercutting drifts are run across the block at 25-foot intervals as shown in Figure 23. These drifts are widened at short intervals to allow the use of long steel in drilling rounds in the intervening pillars. Concurrently with this work the tops of the finger raises are belled out. The pillars are then drilled and shot a section at a time. In the Live Oak division undercutting is done by connecting the finger raises by a series of inclines, as shown in Figures 31 and 34. The intervening pillar between each row of raises is then drilled with stopers and shot a section at a time. This system of undercutting can be used only where the ground stands well.

At Miami the undercutting level is started 30 feet above the grizzly level from the tops of the finger raises, utilizing one or several of them as supply raises and manways and those nearest to the work for drawing off the broken ore. The undercutting level and its relation to the grizzly level, chute sets, and draw points is shown in plan and section in Figure 32. The chute sets are numbered from 1 to 36 and the draw points are lettered. The undercutting level is opened by driving four drifts of small cross section parallel to the grizzly drifts through every third line of draw points, 37 1/2 feet apart and equidistant each side of the central grizzly drift. These drifts are connected at both ends by fringe drifts. Undercutting is started by putting up a shrinkage stope two or three rounds from one of the fringe drifts. This shrinkage stope is usually carried along the side boundaries and one end of the stope as undercutting of the main body progresses. The main body is undercut from drifts 8 feet wide, which are driven at right angles to the small opening-up drifts.

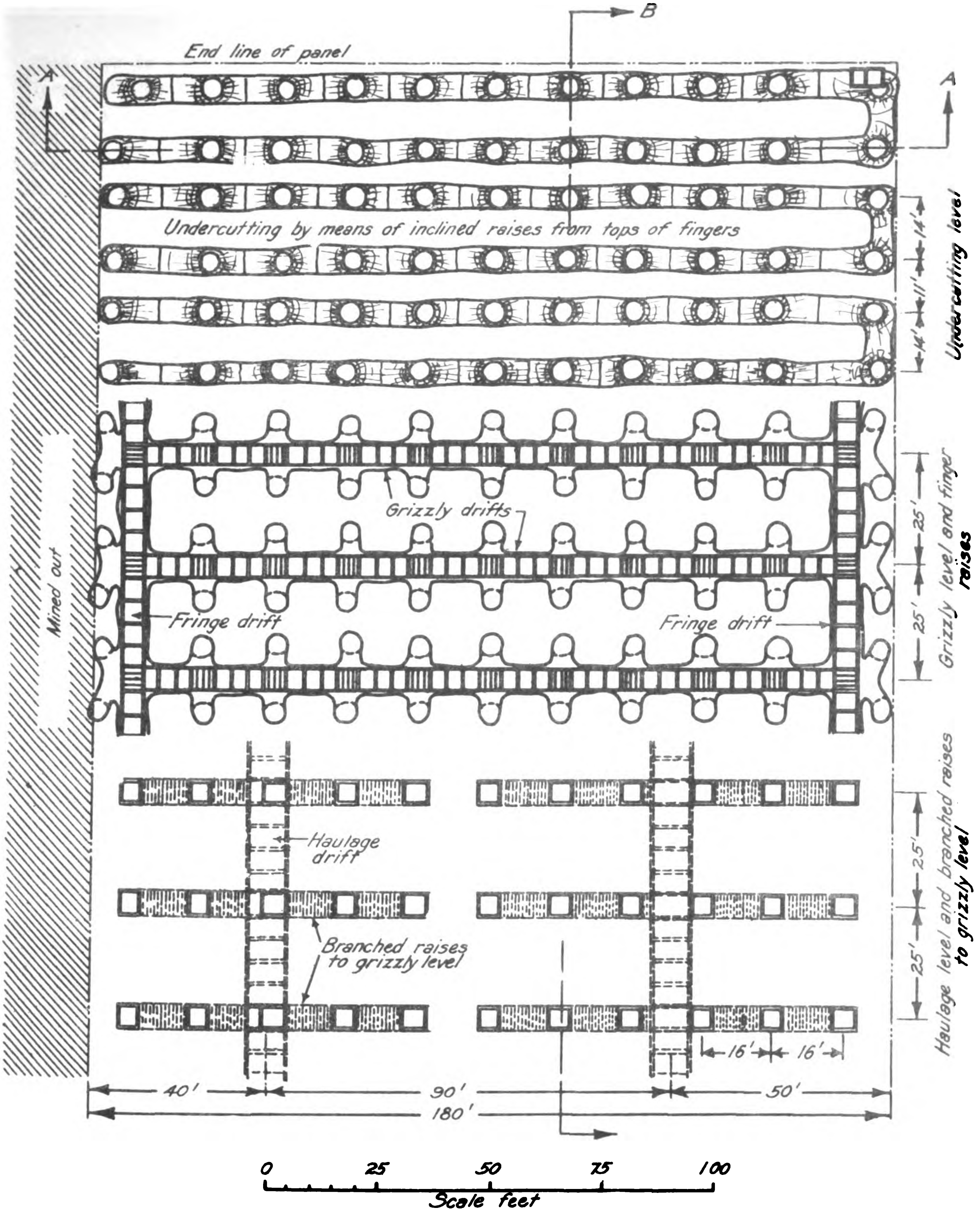

B
End line of panel
A
A
Undercutting by means of inclined raises from tops of fingers
14'
11'
14'
Undercutting level
Grizzly drifts
Fringe drift
Fringe drift
25'
25'
Grizzly level and finger raises
Mined out
Haulage drift
Branched raises to grizzly level
25'
25'
Haulage level and branched raises to grizzly level
16'
16'
40'
90'
50'
180'
0
25
50
75
100
Scale feet

Figure 34.- Plan of haulage, grizzly, and undercutting levels, Live Oak division, Inspiration mine

This work is carried back diagonally, as shown cross hatched on the plan (fig. 32). The large drifts are not driven until needed for blasting, in order to avoid premature crushing. On the plan 9, 10, 19, 18, 29, and 28 represent drifts which are ready to be drilled and blasted, while 7, 17, 16, and 26A are just being driven. The sides and backs of the drifts are drilled and blasted, which completes the undercut. The tops of all finger raises are funneled as undercutting progresses. The undercutting of a stope, once it is started, is completed as rapidly as possible. The small opening-up drifts, however, owing to their small cross section and thick pillars between, may be driven in advance and held without damage until necessary to start undercutting operations, thus promptly providing numerous working faces for the undercutting crews.

Usually ore-drawing at Miami is not started until the stope is completely undercut, although there are exceptions to this rule, as when drawing is started to avoid excessive packing of soft ore and to ease weight on the chute sets and grizzly level below.

At the Copper Queen a gridiron system of drifts and crosscuts 7 to 8 feet high is run at an elevation 19 feet above the bottom of the grizzly level, leaving pillars 15 feet square. Concurrently with this work the tops of the draw raises are belled out (fig. 33). Stub drifts are then run into each pillar to allow the use of long steel, as shown in the figure, and the pillars drilled with a fan-shaped round, which when blasted completely cuts the pillar.

At Ruth five or six series of raises are completed before undercutting is started. A raise series is a row normal to the haulage drift, as shown in Figure 25, A-A. The ore is caved by undercutting from the tops of the finger raises. The tops of the two lines of finger raises in a raise series are connected by drifts, as shown in Figure 25, B-B. Then crosscuts are driven to connect the tops of each alternate pair of finger raises in the other direction. The next step is to drill and blast the pillars remaining between the crosscuts and drifts, which allows the block of ore to cave. In the undercutting workings the openings stand with little timber the few days necessary that they remain open. If the ground is heavy the undercutting operations may consist of rounds of vertical and horizontal holes drilled from the top of each finger raise without running the drifts and crosscuts. Little difficulty is encountered at the Ruth mine in starting the ore to cave.

At the Andes Copper undercutting drifts are spaced 27.3 feet apart and are run 19 feet above the bottom and midway in plan between the drawing chute sets (fig. 26). Crosscuts 15 feet long are run at intervals from the drifts. This is to provide room for handling long steel and thus avoid the expense of supporting a large opening should the ordinary widening of the drift be practiced. Undercutting holes 10 to 13 feet long are drilled from the crosscuts for blasting the pillars. Undercutting is advanced diagonally across the block.

A different system of undercutting from that used at the other mines is used at Ray. It is a development of the original stope-and-pillar method of mining and consists of undercutting the block by means of shrinkage stopes (fig. 16). The grizzly drifts are 25 feet apart. Two "finger" or "throat" raises are put up from each grizzly and inclined so as to be 12-1/2 feet center-to-center. Over alternate rows of throat raises—that is, at 25—

foot intervals--the first series of shrinkage stopes, called locally "undercut stopes," is mined up to about 43 feet above the grizzly level. These stopes extend from side to side of the block. They are about 6 feet wide, except that the final round is fanned out to cover a width of 17 feet. Between the undercut stopes, directly over the remaining rows of throat raises, similar but smaller stopes are begun. These are carried to a height of 35 feet above the grizzly level. The final round is likewise drilled as wide as possible, and when blasted breaks into the stope on either side, completely cutting off the pillar and allowing the ground above to cave. Both undercut and pillar stopes connect with their final round at either end with the side cut-off stopes.

The method of undercutting used at Ray has been adopted for two reasons: First, to provide pyramids or cones of solid rock above the drawing level, which tend to support the broken ground and keep the direct weight from coming upon the draw sets or upon the drawing level. Second, this method of undercutting breaks approximately 43 feet of ground over the draw point. This considerable thickness of broken ore not only acts as a cushion, which protects the grizzly level from concentrated pressures, but as it is drawn off it allows more thorough breaking up of the main mass of ore before the latter reaches the draw points.

At the Old Dominion, where the Morenci slide system is used (fig. 28), the first operation in undercutting is to cut the raises for draw holes. These raises apex midway between the slides. The apex between two draw raises from adjoining slides is left higher than the top of the timber of the slides in order that the pillar thus formed will bear part of the weight of the overlying broken ore. A cut of at least 4 feet is then taken out directly over the length of the incline. This initial cut over the inclines is important. There must be enough room to accommodate the swell from the first slough of ore so that no direct weight is thrown upon the timbers.

The next step is to blast another round in the back of the draw holes. Finally the pillars between the draw holes are drilled and blasted. A pillar from apex to apex is blasted down as a unit and must always be entirely cut in order to reduce the probability of a large section of unbroken ground settling on the slides.

A draw hole from the next slide beyond the pillar being shot down should be completed before blasting. This gives a safe entrance from which to observe the effect of the blast. In many cases it is possible to bar down, drill, and blast the back a second time. This would be impossible were this safe entrance not prepared in advance.

As a safety measure in undercutting no holes may be drilled in a pillar beyond a vertical plane separating two pillars, to prevent drilling into missed holes. The undercutting of pillars starts from the footwall side of the vein and progresses to the hanging wall.

The amount of ore broken in development work in side cut-off workings and in undercutting at the Morenci, Inspiration, Miami, and Copper Queen mines amounts to about 10 per cent of the total ore in an average block 200 feet high. Increasing the cut-off workings at Miami from 25 to 50 per cent of the area around the block would increase the total amount of ore broken by drilling and blasting to about 14 per cent of the total. At the Copper Queen a complete cut-off around an average block would increase the amount broken by blasting to about 19 per cent.

At Ray, with a different method of undercutting, about 20 per cent of the ore in a block with a 200-foot lift is broken with explosives.

DRAWING PRACTICE

The two main objectives in drawing ore when a caving method of mining is used are: (1) To draw a maximum of the ore tonnage with a minimum dilution by waste; and (2) to regulate the drawing to prevent damaging weight on the extraction workings below the ore if possible, or to relieve such weight if it occurs for maintenance and ore-drawing costs to be kept at a minimum and for minimum interference with the predetermined order of drawing. Ore drawing as it actually works out in practice is a compromise between the requirements of these two principal objectives.

After undercutting the ore is drawn slowly at first until it starts to cave freely and afterward at the rate that best suits the nature of the ground and the requirements for ore. After starting, drawing should be continuous to avoid packing of the ore. Moreover, pressure on the grizzly drifts is affected by the rate of drawing. If it is too slow, weight on the grizzly drifts will gradually accumulate, while rapid drawing relieves that pressure. Too rapid drawing, however, may bring down the ore to the chutes in too large fragments to be economically handled.

The proper rate of drawing depends on the rate at which the unbroken ore caves. If the ore is drawn too rapidly too large an open space will appear between the broken and unbroken ore. In such case the ore is likely to break in large masses. If it is drawn too slowly the swell of the broken ore will partly support the arch above and delay caving.

After caving reaches the top of the ore there appears to be no limit of drawing speed other than the ability to put the ore through the grizzlies and preserve a proper relative movement of the ore and waste capping to the chutes.

A uniform draw, particularly during the early stages of drawing, is essential for satisfactory recovery. The contact between the ore and broken overlying capping should be an even plane, whether horizontal or inclined. If the early drawing is not done uniformly the mass of ore may cave along some unknown plane of weakness, causing chimneys or cavities which may extend into the capping and allow it to mix with the ore. After the main mass of ore has been thoroughly broken and the block has caved to the surface the result of subsequent irregular drawing is not as serious as would have been the case earlier.

In drawing the ore is worked through the draw points onto the grizzlies, where fragments too large to go through are broken either by hand or by blockholing. No bulldozing is done on the grizzlies because of possible damage to the timber of the grizzly sets. At Ray and at some of the other mines no blasting is done on the grizzlies for this reason.

If the ore is well broken in caving it can be started to run in the draw points by barring. Occasionally a large bowlder will block a draw raise; in this case it is drilled and blasted. In hard-rock mines the raises are funneled to get such large fragments as near the grizzly level as possible, so that men will not have to go into the raises to drill holes. At all mines the draw raises will hang up occasionally because of large bowlders arching over the raise or ore packing between drawing periods. Such hung-up chutes as can not be brought down by barring are started by blasting with charges of dynamite, called bombs. The bombs, of 4 to 10 cartridges tied into a bundle, are fastened to blasting sticks and placed in position under the hung-up ore and shot. Detonating such charges with electric detonators is preferable to shooting them with fuse and caps. By the use of electric firing the chute tapper can place his bombs in the most favorable positions in the hung-up raises, is not required to work in a smoky atmosphere as when using fuse, and has ample time for all operations, so that he is not hurried and does not have to work under strain. Also, men are less likely to be too near when a shot is fired, there is less delay in waiting for missed shots, and there is no danger of a bomb already spit falling down a raise to the haulage level. To prevent accidents to the men drawing the storage raises on the haulage level an electric signal system is used at most mines to give warning of blasting to be done on the grizzly level.

At Morenci the drawing practice is governed by the presence of a large proportion of hard, coarsely-breaking ore, as a result of which three innovations have been developed: (1) Drawing through large openings in the sides of the grizzly drifts instead of through timbered chutes; (2) using grizzlies with bars spaced 16 inches apart instead of the customary 10 or 12 inches; and (3) using one man to draw the ore and also work it through the grizzly.

So much coarse rock occurs in every section of the mine that ore can not be drawn through small chute doors satisfactorily. Openings 4 by 6 feet in section are therefore cut in the sides of the grizzly drifts to connect with the finger raises. The ore flows through these large openings to the grizzlies where it is broken up by sledging or blockholing. At the grizzlies there is enough room for a man to work and to get away if a surge of ore occurs. Wide spacing of grizzly bars reduces sledging and blockholing, although it then becomes necessary to change haulage, hoisting, and coarse-crushing equipment to accommodate large bowlders. The advantage of the wide spacing is reflected in chute-tapping efficiency, which has increased to its present figure of 120 tons per man shift from an average of 37 tons per man-shift with 10-inch grizzlies. As a protection against falling through widely-spaced grizzlies the chute tapper is provided with a safety belt which he always wears while at work on the grizzly. Contrary to the popular impression before this innovation was tried the belt does not interfere with the output of the workman. When not actively in service all grizzlies are covered with two 10-inch planks. Handholds of substantial chain are installed on both sides of each grizzly.

Previous to the introduction of the wide grizzlies it was occasionally impossible to put the ore through as fast as stope conditions dictated. Drawing can now be increased to a rate of 1,200 tons per day from an area of 10,000 square feet whenever it may be found desirable to do so; this means caving at a rate of 1-1/2 feet vertically per day.

In the active portion of a producing panel some ore must be drawn daily from each finger to prevent local packing. At Morenci the minimum draw in areas which are being held back for purposes of stope control is 12 tons per finger.

At Ray individual blocks are drawn separately, as each block is cut off on all sides by cut-off stopes or finished sections. Undercutting operations are carried on progressively from one end of the block to the other and are immediately followed by drawing operations on a limited scale, to make space for the expansion of the ground above as it caves. This gradually removes the support from under the main mass of ore and by setting up a cantilever action causes it to slough and break up. As soon as the block is completely undercut the regular procedure of uniform drawing is started throughout the block. The same procedure is followed in the next block of the panel, undercutting and drawing being started at the end remote from the previously-mined block.

At Miami, immediately after undercutting of the stope is completed ore drawing is started. It is usually slow work at the beginning, as the first ground caved generally arrives at the chutes in fairly large pieces. Sometimes the stope back hangs up and drops only a very little ore for several weeks, but this is unusual. To maintain the normal daily production of approximately 18,000 tons of ore it is necessary to have 13 or 14 stopes on the drawing schedule. Of this number the stope or stopes about to be finished and those just beginning produce a comparatively small tonnage. In addition to the ore produced from actual ore-drawing operations there is a small tonnage produced from stopes in preparation for ore drawing, and from mine development.

In one month's operation, 17,185 tons were produced daily from ore-drawing operations in 13 stopes, which was equivalent to an average of 1,322 tons per stope, or 17.06 square feet of area required per ton of ore produced daily. The corresponding average thickness of solid ore drawn daily would be 0.733 foot.

The chute sets are numbered from 1 to 36, inclusive, and the four chutes in each set are lettered A, B, C, and D. The A chutes are drawn by one shift, B chutes by the next shift, and so on in rotation. The drawing is done by crews of two men in each, consisting of a chute blaster, who receives miner's wages, and a chute tapper, who receives mucker's wages. The chute blaster draws the ore from the chute set and the tapper keeps the grizzly clear on the grizzly level. The blaster does all ordinary chute blasting required, but in addition to this there is a special crew of chute blasters and chute drillers who bring down especially difficult chutes or hung-up raises. All chute blasting except the blasting of loaded drill holes is done by means of small hand batteries.

The tapper crew will usually draw from 12 to 15 finger raises per shift, blasting five to eight times and drawing approximately 400 tons per shift, Normally 50 tons is drawn daily from each chute listed on the draw orders. The aim is to draw this amount in rotation from each chute in the stope, but variation in this regular routine is brought about by various causes, such as the presence of capping in a chute; necessity for repairs; presence of weight on the timbers, requiring the chutes to be drawn to relieve this weight whether listed to be drawn or not; variation in drawing to meet requirements as to grade of ore; and changes in order to drawing to maintain proper distribution to haulage level for economical operation of the trains.

Drawing is continued from each chute until the grade of the ore drops to a point that is not sufficiently profitable. This usually dous not occur until after all of the expected ore has been drawn. This cut-off point is fixed mainly by the minimum profit required, which may vary, depending on the cost of operation in the particular stope, market price of the metal, capacity of treatment plant, and total profit requirements. It is also affected by whether there is another lift of ore to be drawn from beneath it or not. In case there is, the grade of the ore at which drawing stops may be higher, as there will be another opportunity to recover it from below.

Although the large capacity of the transfer raises makes ore drawing fairly independent of transportation on the haulage level below the method of measuring ore drawn requires close correlation between these two operations; this is maintained by telephone service between the draw bosses and the train dispatcher on the haulage level, who issues orders to all trains on their return trips from the shaft.

The Ruth ore tends to chimney directly over the draw points. These chimneys will sometimes stope themselves to the capping, which allows the capping to appear at the draw points, leaving a pillar of ore directly over the draw set. When the raises chimney up in this manner caving is started in the pillar of ore by drilling holes around the base of the pillar from consecutive raises. By using the 12 1/2-foot spacing it is always possible to start caving of the pillar when the ore chimneys, thus insuring maximum recovery of ore. If greater spacing of raises were used the pillar of ore could not be attacked without danger and considerable expense. Better extraction is obtained by drawing slowly over a large area whenever possible rather than by drawing hurriedly from a small block, as the plane of contact between the ore and capping is kept at a more uniform angle. In drawing over a large area, however, the broken ore must stand for a longer time before it is won. The longer the ore stands the greater the tendency for it to pack and hang up, and the maintenance costs are increased; so a medium is struck between the opposing factors.

Experience enables the mine foreman or his assistants to judge from the condition of the ore the best practice to follow in each case.

At Inspiration in working blocks as separate units the area farthest from previously-caved ground is undercut first, and drawing is begun. If two sides of the block have been previously mined the undercutting is done on a diagonal. About 40 per cent of the block is drawn by the time undercutting is completed for the block. By this procedure it is considered that dilution from the completed stopes is kept to a minimum.

Drawing is done from as few lines of grizzlies as possible and still maintain production.

Methods of Controlling Drawing

For maximum recovery of the ore and minimum dilution the ore must be drawn uniformly. To obtain this end the drawing of the chutes must be closely controlled. Occasionally a number of chutes will continually hang up during a shift, which makes it difficult to draw the required daily tonnage for the concentrator. At such times when production is being pushed there is a tendency to pull ore from free-running chutes at the expense of grade or ultimate recovery of the ore.

At most mines the control of drawing is under the engineering department. Chute checkers reporting to the engineering department supervise the drawing. When the required quantity for the day is drawn from a chute it is sealed. The seals are not broken until it is time to draw from the chute again. Also, when a chute begins to run waste it is sealed. As the capping is usually of a different color than the ore waste can easily be detected.

At Morenci when a new stope is ready for production a plan and section are prepared by the stope-control department on a scale of 20 feet to the inch to assist in controlling the drawing of the ore. The plan is a special map of the stope showing each finger or draw raise and the transfer raise into which it empties. The line of fingers is numbered in one direction to show the page number on the section; in the other direction the numbers designate the grizzly lines (fig. 35). Thus "4-96" is used to designate the first finger on the south side of the bottom row. This method identifies the finger on the office records and also gives its location underground. After the plan is completed the tonnage tributary to each finger is calculated from the ore outline shown on the assay maps and adjusted to agree with estimates of ore reserves. These tonnages are recorded on the finger map in the rectangles shown in black in Figure 35, but which are white on the blue prints. Longitudinal sections are then made by plotting these tonnages to scale. Each page of sections represents a line of fingers. These sections are based on broken ore volumes as distinguished from rock in place. The ore in place runs 12 cubic feet to the ton, while the broken rock is considered to be 20 cubic feet to the ton, so that on the sections each finger is increased 40 per cent to allow for the breaking up of the rock. The increase in volume is termed the swell. In the present stopes, where the fingers are spaced on 14-foot centers in one direction and 18 2/3-foot centers in the other, each foot in height represents 13.07 tons of ore. The tonnage is posted on the sections in different colors according to the month the ore was drawn.

Drawing operations in the stopes are controlled primarily from the sectional maps. They show (1) the angle of retreat, (2) when a finger is being pulled too fast, (3) when a finger is not being pulled at all, and (4) when the stope as a whole is being pulled too slowly or too rapidly.

If a pipe of cap rock reaches a draw point in advance of the upper boundary of the ore body the finger is sealed, and the ore is drawn from the surrounding raises until the waste pipe is broken. Close spacing of draw points facilitates the breaking up of pipes of waste; if the draw points are too far apart they do not assist each other.

When the ore in a section is nearly all drawn off the rate of drawing is decreased to give time for sampling doubtful chutes and obtaining the maximum recovery of clean ore.

A chute checker whose duty is to see that the ore is drawn according to the draw sheet which is issued to him each day by the control department is employed on each shift for each stope. He makes a tonnage report for each finger worked during his shift and also takes samples from each finger. He turns in a report from the transportation department showing the number of cars drawn from his stope during the shift. He closes off the fingers marked "waste" and also makes a note of any new fingers that start running waste during his shift and reports this to the office. The chute checker keeps track of the haulage chutes that are full and delivers a list of these to the transportation crew at regular intervals.

When the reports are received in the office the cars are apportioned according to the fingers worked. The tonnages are then posted on the finger sections, and new draw sheets are made out. The draw sheet shows the total number of tons released in each finger, the fingers to be closed off, and the maximum daily rate of draw for each finger. The results from the finger samples are posted on a sheet showing the number of cars drawn from that finger during the day. These assays are all averaged together according to their tonnages to give a composite result for the entire stope production for the day.

In the Ray mines stope control is also under the engineer department and is directly supervised by the stope engineer, who lays out stoping plans and calculates the tonnages and assay values of blocks. Under him are stope-control engineers, one on each shift in every working block. It is their duty to see that an even draw is maintained in the drawing section, to take grab samples of the ore at the draw sets, to tally cars on daily written reports, and to seal draw points in case of waste or overdrawing. Chutes are sealed by cardboard strips with the initials of the sealer being tacked to the chute boards so that the gate can not be reopened without breaking the seal.

Composite assays for each drawing lateral for each shift are posted daily at the mine and at the engineering office. These are for the guidance of the stope engineer, as well as the mine bosses. Individual draw-point assays are also posted. In each block a chart shows the number of cars drawn to date from each draw point and is brought up-to-date at least twice each week. This serves the foreman and bosses by indicating the condition of the draw. A similar chart in the engineering office shows cars drawn to date from each draw point, and in addition shows the original expectancy for each draw point.

The ore drawing at Miami is under the supervision of three stope engineers on day shift, who inspect the stopes daily and issue ore-drawing orders to four draw bosses and five chute sealers on each of the three shifts. From these orders the draw bosses issue orders to the different crews under their direction, listing the chutes to be drawn in groups of chutes which empty into the same transfer raise.

Each stope engineer inspects daily all chutes listed on the draw sheet and receives reports from draw bosses and sealers. He notes the condition of the ore in all the chutes, removes from the draw list chutes in which capping shows, lists chutes that need drilling or blasting and chutes or chute sets or grizzly level timbering that require repair, and orders this done in the proper sequence.

Any chute in which capping shows up prematurely is sealed for several weeks before being placed on the drawing list again. In the meantime the chutes surrounding it may be sealed and later drawn, which tends to break up the wall of the pipe, after which the chute with capping may usually be drawn again with satisfactory results.

The stope engineer carries in his notebook graphs representing the tonnages drawn from the chutes along each row. These are posted every two days so that as he inspects the stopes he has reference to an accurate record of the stopes' condition. Office records of each stope are kept. These consist of similar graphs of the tonnages drawn along each row of finger raises; there are twelve graphs for each stope, which are posted semi-monthly.

Wooden 12 by 12 inch marker blocks with brass number tags are planted at 25-foot intervals around all the boundary caving drifts and in such other locations in the stope as are accessible on the boundary caving levels. The brass tags are marked with the number of the stope, the level on which the block was planted, and its location. When, during the drawing of the stope, the blocks come out on the grizzly level, the square set and chute through which each block comes is noted. The position from which the ore is coming can thus be ascertained whenever a block is recovered. The places where the blocks are planted and the places where they come out on the grizzly level are platted. If the blocks drop vertically it is evident that the ore is being drawn correctly; that is, each finger raise is drawing the ore directly above it. Where a block shows much horizontal movement the finger raise through which it is recovered is drawing ore that should come through another raise, and the drawing is corrected accordingly.

To draw the ore down evenly it is necessary to have a method of measuring the ore drawn and of recording this so that there is a continuous record to indicate the chutes that should be drawn next. The method of ore measurement and control now in use utilizes the ore cars on the haulage level as units of measurement; but as 24 finger raises discharge into each transfer raise and as it is essential to record the tonnage that comes from each individual finger raise, it is obvious that in order to utilize the ore cars as measuring units the finger-raise drawing and transfer-raise drawing must be coordinated. This is brought about by listing the chutes to be drawn by the chute tappers in such a way that they draw one chute only which empties into one transfer raise and so on with succeeding transfer raises, and another chute emptying into the first transfer raise is not drawn until this raise has been drawn on the haulage level. By coordinating the time of drawing the finger raises with that of drawing the transfer raises, an exact measurement of the ore drawn from each finger is obtained for about 75 per cent of the ore drawn.

At Ruth, close supervision of drawing is required to prevent dilution and to obtain a maximum recovery of ore. Charts showing vertical sections through the plane of each series of raises are kept up-to-date; moreover, a graphic representation of the ore above each finger raise, and the relation of the capping to the top of the ore in drawing, is provided by means of a model. This model consists of 1/8-inch steel rods set vertically in a wooden base. Each rod represents a finger raise or draw point. The scale used on the model is the same as in the charts. To each steel rod is attached two steel disks, 1 inch in diameter and 1/2 inch in thickness, which are adjustable on the rods by means of set screws in the disks. The lower disk represents the bottom of the ore body and remains fixed; the upper collar representing the top of the ore body at each finger raise is lowered as the ore is drawn. The adjustable collars are placed to scale from the vertical charts. The charts and model are kept up to date by the stope engineer.

In drawing the ore from the finger raises records are kept by the draw bosses of the amount of ore drawn, as estimated by the tonnage run into the branch raises. As the branch raises are of standard dimensions the draw bosses become adept at estimating the tonnages. A record is also kept by the motorman of the quantity of ore drawn from each branch raise. At the end of the shift any discrepancy between the two reports is adjusted by the stope engineer. At the end of each week the tonnages as estimated above are corrected to conform with the actual tonnages delivered to the concentrator. These corrected tonnages are posted on the charts and the upper collars on the steel rods are lowered in proportion to the amount of ore withdrawn for the week from each particular finger raise. The "swell" or increase of volume of the ore when broken is neglected.

The stope engineer prepares for the mine foreman each evening a report which shows tonnage to be drawn from each finger raise the next day.

ORE RECOVERY AND DILUTION

As previously stated, it is not possible to obtain all the ore in a block by a caving method, or to prevent some dilution with waste. Recovery is usually calculated both in tonnage and in metal content. Tonnage recovery is the ratio of the tonnage drawn from a block (extraction) to the tonnage estimated as being originally contained in the block (expectancy). Metal recovery is the ratio of pounds of metal drawn to pounds of copper expected.

The accuracy of the recovery calculations depends upon the accuracy of the sampling and estimation of the ore in the block, as well as the record of the grade and tonnage drawn from the block. An apparent extraction of over 100 per cent of the ore and of nearly 100 per cent of the copper is frequently obtained. This may be due to an underestimate of the grade and tonnage in the block or to disregarding copper in the capping or other material with which the ore is diluted. For instance, if 1 per cent copper is the cut-off point for profitable ore and considerable capping is pulled that runs 0.8 per cent copper, the copper in the diluting material may equal the copper contained in the ore lost. At most mines all faces around a block are carefully sampled, and a correction is made for grade and tonnage after all development work in the stope preparatory to drawing has been done. A more reliable calculation of recoveries can therefore be made, using such figures, than when the estimate of tonnage and grade is based upon the original drillhole sampling.

At Ray in mining over 40,000,000 tons from completed sections records indicate that the dilution experienced to date approximates 10 per cent.

At Inspiration, where adequate plant capacity is available, ore reserves are figured to include material down to a point where the margin of profit in mining ceases. Under these conditions a minimum efficiency of 75 per cent is figured, efficiency being here defined as percentage copper recovery divided by percentage tonnage recovery. The recovery of the estimated tonnage is in excess of 110 per cent, with a recovery of over 85 per cent in metal content.

At Miami, Maclennan[16] reports:

> Up to date 13 stopes have been completely drawn. Nine of these were original stopes surrounded on all four sides by solid ground and four were pillar stopes, two of which were adjoined by broken waste on two sides and one end, and two adjoined waste on one side and one end. These 13 stopes were estimated to contain 11,038,070 tons of ore assaying 1.026 per cent copper. They produced 12,710,378 tons, assaying 0.912 per cent copper. The table gives a tabulation showing the tonnage, grade and copper extraction from these 13 stopes, the same figures for the best and for the poorest original stope, and for the best and for the poorest pillar stope.

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[16] - See footnote 13.

Tonnage and Grade Extraction

(Partitions not included in expectancy)

	Expectancy		Mined		Percentage extraction		
	Tons	Copper, per cent	Tons	Copper, per cent	Tonnage, per cent	Grade, per cent	Copper, per cent
Total of 13 completed stopes	11,038,070	1.0260	12,710,378	0.9124	115.15	88.93	102.40
Best original stope (2)	998,016	1.0388	1,210,424	1.0091	121.28	97.14	117.81
Best pillar stope (11)	319,560	1.0640	387,827	.9348	121.36	87.86	106.63
Poorest original stopes (7)	1,071,535	.8701	1,053,153	.7786	98.28	89.48	87.94
Poorest pillar stope (9)	1,098,313	1.1067	1,025,032	.8995	93.33	81.28	75.86

The extraction of more than 100 per cent of the estimated copper content of the ore may be due to drawing some of the partition ore, and to copper not included in the estimate, coming from overlying capping or gob.

The best pillar stope, 11, was bordered by waste on one end and one side, and the extraction from it compares favorably with the best original stope, the tonnage extraction being almost identical, while the grade extraction is 10 per cent lower. The poorest pillar stope, No. 9, is the poorest of the lot, but it was the first pillar stope mined. It was 150 by 300 feet in plan, and so much weight developed during its extraction that results were far from good. The experience gained here resulted in reducing the size of the stopes to one-half of their original size. No. 11, the best pillar stope, was a one-half size stope, and was bordered by waste on one side and one end.

COSTS

The cost per ton of mining ore by a caving method, though usually higher, compares favorably with the costs obtained at open-cut mines.

The actual mining cost at the Utah Copper—the largest of the open-cut copper mines—for 1928 was $.1173 per ton of ore. Stripping the over-burden cost $0.2737 per cubic yard.[17] The ratio of stripping to ore is 1.1 to 1.

The total cost of steam-shovel mining exclusive of taxes and depreciation at the Nevada mines of the Nevada Consolidated Copper Co. for 1929 was $0.2415 per ton of ore and $.6232 per cubic yard of overburden.[18] An added charge of $0.40 per ton is added for prepaid stripping. The stripping ratio is 1 to 1.

The total cost of open-cut mining exclusive of taxes and depreciation at the Chino mine of the Nevada Consolidated was $0.3138 per ton of ore and $.4961 per cubic yard of overburden.[19] To this is added deferred charges of $.45 per ton. The ratio of stripping to copper is 1 to 1.

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17 - Soderberg, A., Mining Methods and Costs at the Utah Copper Co.: Inf. Cir. 6234, Bureau of Mines.

18 - Annual Report for 1929, Nevada Consolidated Copper Co.

19 - See footnote 18.

The lowest underground mining cost reported at a large-scale nonferrous mine for 1928 is at the Alaska Juneau, where a sublevel caving method is used. The underground cost per ton of ore drawn was $0.2966.

The cost per ton of ore mined at the Ruth mine of the Nevada Consolidated Copper Co. was $0.95 per ton. To this is added deferred charges of $0.40 per ton.

A mining cost of $0.39937 per ton is reported by the Miami Copper Co.,[20] as shown in Table 3.

Table 3 - General cost data at four copper mines using block-caving method.

	Morenci	Miami	Ray	Inspiration
Period	1928	4 yrs. av.	1928	1928
Tons mines	1,483,984	4,134,074	3,243,159	4,897,646
Tons per man in stopes	62.76	—	115.61	—
Tons per man total underground	11.84	27	12.13	13.1
Tons per man total on payroll	10.45	—	10.83	—
Pounds explosives per ton	0.19	0.223	0.319	—
Timber, board feet per ton	0.25	1.045	1.88	—
Power in k.w. hrs. per ton	—	1.9	—	—
Copper in ore, pounds per ton	35	11.4	21.3	18.3
Mining cost per ton	$ 0.356b	$ 0.399	$ 0.677c	$ 0.583a
Mining cost per pound copper	0.010	$ 0.035	$ 0.033	$ 0.032

a - Partial underground costs, 1928.
b - Extraction cost only.
c - The total underground cost exclusive of taxes and depletion for 1929 was $ .8275 per ton.

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20 - See footnote 13.

Table 4 - Mining costs during four-year period (Oct. 1, 1925 to Sept. 30, 1929) at Miami Copper Co.

Tons mined: 16,556,296.

	Cost per ton
Development	$0.10000
Stoping	.13628
Electric haulage	.05314
Hoisting	.03317
Ventilation, etc.	.02082
General underground	.01149
Engineering and sampling	.01382
Mine surface	.01979
Mine accident	.01086
Total	0.39937
Average tons mined per shift underground	27
Timber consumed per ton of ore	1.045 bd. ft.
Explosives (40 per cent gelatin dynamite) used per ton of ore	.2225 pound
Tons mined per pound powder	4.5
Power required in mining, per ton of ore mined, kw.h. (slightly more than 1 kw.h. used for hoisting).	1.9

At the Ray mines of the Nevada Consolidated Copper Co. the total underground cost for 1929, exclusive of taxes and depreciation, was $0.8275 per ton. To this is added $0.15 per ton deferred development charges.[21]

The following tables give the mining costs for 1928 in detail.[22]

Table 5. - Summary of costs, year, 1928, Ray mines

	Labor	Timber	Explosives	Power	Machine	Misc.	Total
Preparatory excavation	$0.0581	$0.0339	$0.0070	$0.0014	$0.0010	$0.0022	$0.1036
Stoping	.0711	.0082	.0247	.0050	.0016	.0059	1.165
Reserve drawing	.0940	.0129	.0190	.0001	.0001	.0006	.1267
Haulage (including operation tipples)	.0585	.0006	—	.0078	.0139	.0101	.0909
Hoisting	.0084	.0001	—	.0071	.0014	.0027	.0197
Misc. mine expense	.0368	.0001	—	.0114	.0016	.0074	.0573
Plant administration and general	.0445	.0005	—	.0023	.0018	.0602	.1093
General overhead (exclusuve of taxes)	—	—	—	—	—	.0526	.0526
Totals	0.3714	0.0563	0.0507	0.0351	0.0214	0.1417	0.6766

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[21] - See footnote 18.

[22] - Thomas, Robert W., Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz.: Inf. Circu. 6167, Bureau of Mines.

Table 6. - Labor, powder, and timber consumption, 1928

Year	Tons mined	Tons per man			Powder per ton, pound	Board feet per ton
		Stoping	Mine pay roll	Over-all		
1927	2,551,913	101.60	10.66	9.39	0.2523	1.79
1928	3,243,159	[1]115.61	12.13	10.83	.3193	1.88

[1] - This figure estimated because of change in cost charges between the years 1927 and 1928.

Table 7. - Labor, powder, and timber consumption—development, 1928, Ray mines

	Size of heading, feet	Feet per man-shift	Powder per foot, pounds	Timber per foot, board feet
Grizzly drifts (untimbered)	4 by 5	2.16	4.84	—
Supply drifts (timbered)	6 by 8	1.16	5.39	50
Motor drifts (timbered)	8-1/2 by 9	.84	6.92	113
Averages, total driving and timbering	—	1.05	—	—
Throat raises	4 ft. dia.	9.2	4.17	—
Short haulage raises (untimbered)	4 by 4	5.67	4.72	—
Cribbed raises	5 by 5	2.58	4.90	8
Average raises	—	4.50	—	—
Grand averages, all driving and timbering	—	1.38	5.69	52.38

Table 7 shows the percentage of ore mined by each method and obtained by development work during the year.

Table 8. - Production of ore by each variation of the mining method and by preparatory excavation at Ray during year 1928

	Tons	Per cent of total
Preparatory excavation	186,742	5.76
Hand-tramming method	119,981	3.70
Regular motor method	388,154	11.97
Modified motor method	294,769	9.09
Sub-level method	2,253,513	69.48
Total mine production	3,243,159	100.00

The following table gives the summary of partial underground costs at the Inspiration Consolidated Copper Co. for 1928.[23]

Table 9. - Summary of costs, Inspiration Copper Co., 1928
Tons of ore hoisted: 4,879,646.

	Partial underground costs per ton ore hoisted							
	(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)
	Labor	Supervision	Comp. air, drills and steel	Power cost	Explosives	Timber	Other supplies	Totals of partial costs
Development	$0.123	$0.007	$0.019	—	$0.016	$0.040	$0.006	$0.211
Mining	.146	.007	.005	—	.017	.031	.002	.208
Transportation	.111	—	—	$0.014	.003	.013	.003	.144
Gen. underground expense	.012	—	—	—	—	—	—	.012
Surface expense	.008	—	—	—	—	—	—	.008
Totals	.400	.014	.024	.014	.036	.084	.011	.583

Notes:

Item (3) includes power for operating drills, labor sharpening drills, etc., but labor for operating drills is included with Labor shown in (1).

Item (4) includes all power used for haulage underground and hoisting only. These two items, with air compression, comprise the greater part of power used. Other items of power negligible in terms of cost per ton of ore hoisted.

Items (5), (6), and (7) include labor and sundry other expenses incident to the handling of explosives, timber, and other supplies.

The direct mining costs at the Morenci mine of the Phelps Dodge Corporation for 1928, exclusive of compressed air and transportation, are shown in Table 10.[24]

The summary of costs of mining in units of labor and supplies at Morenci for 1927 and 1928 is shown in Table 11.

The output per man for the year 1928 on production and work associated with it, including all shifts on the mine payroll with salaried employees, is approximately 18.5 tons. This excludes all lower-level development which is for the purpose of bringing into production the next lower lift.

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[23] - Stoddard, Alfred C., Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.: Inf. Circ. 6169, Bureau of Mines.

[24] - Mosier, McNenry, and Sherman, Gerald, Mining Practice at Morenci Branch, Phelps Dodge Corp., Morenci, Ariz.: Inf. Circ. 6107, Bureau of Mines.

Table 10. - Mining Costs, Phelps Dodge Corporation- Morenci Branch, 1928

	Tons	Cost per ton			
		Labor	Explosives	Timber	Total
Stope Preparation:					
Drifts, driving, timbering, and tracks	—	$0.014	$0.004	$0.006	$0.024
Raises, includes grizzlies	—	.015	.005	.015	.035
General Development:					
Drifts, driving (haulage drifts)	—	.008	.002	—	.010
Drifts, timbering do do	—	.010	.001	.014	.025
Drifts, tracks	—	.003	—	—	.003
Raises, includes chute fronts	—	.018	.005	.012	.035
Underground tunnel and chute repairs	—	.008	—	.002	.010
Stoping:					
Breaking ground[1] and chute tapping	—	.074	.034	—	108
Stope repairs	—	.011	—	.010	.021
Total direct	1,548,258	.161	.051	.059	.271
Average cost per ton -					
Compressed air, drills, tool sharpening, steel, water and sundry supplies (including handling and delivery)	—	—	—	—	.045
Supervision	—	—	—	—	.040
Total extracting ore					.356

1 - Includes undercutting and shrinkage stopes.

2 - All costs based on total tonnage.

Table 11. - Summary of costs in units of labor and supplies, Morenci, 1927-1928

Tons of ore hoisted: 140,000 per month

MINING

Year	Tons mined	Tons per man			Powder per ton, pound	Timber per ton, board foot
		Stoping[1]	Mine payroll[2]	Over-all[3]		
1927	1,136,339	40.35	10.00	9.07	0.24	0.23
1928	1,483,984	62.76	11.84	10.45	.19	.25

1 - Includes: (1) Boundary shrinkage stopes; (2) undercutting; (3) belling fingers; (4) chute tapping; and (5) stope repairs, but is exclusive of all development work.

2 - Tons per man for shifts on mine pay roll, including men on salaries.

3 - Tons per man for shifts charged to mining, including all labor from other departments, such as mechanical, electrical, carpenter, and other surface departments occupied on work chargeable to mining.

The development costs in units of labor and supplies are shown in Table 12.

Table 12. - Summary of Development costs in units of labor and supplies, Morenci, 1927-1928

Tons of ore hoisted: 140,000 per month.

DEVELOPMENT

	Size of heading, feet	Feet per man-shift		Powder per foot, pounds		Timber per foot, board feet	
		1928	1927	1928	1927	1928	1927
Grizzly drifts (raw)	4 by 6)	2.81	2.58	5.12	5.85	—	—
Supply drifts (raw)	6 by 8)			6.43	6.13	—	—
Motor drifts (raw)	8 by 9)	.98	1.26	10.87	8.01	—	—
Motor drifts (timbered	10 by10			9.37	7.50	—	—
Averages, total driving		1.52	1.93	—	—	—	—
Timbering drifts	6 by 8 or less	—	—	.82	.32	58.44	52.94
Timbering drifts	8 by 9 or over	—	—	1.08	.63	107.88	100.98
Averages, total driving and timbering		1.00	1.36	—	—	—	—
Finger raises (raw)	4½ by 4½)	5.71	4.84	3.82	4.83	—	—
Short haulage raises (raw)	4 by 6)			3.97	3.94	—	—
Long raises with dividers	4 by 7	1.47	1.49	5.97	8.25	28.18	16.54
Cribbed raises	4 by 7	1.10	1.73	4.83	2.96	84.20	62.71
Morenci slides	—	—	1.81	—	5.15	—	74.79
Averages, raises		3.91	3.48	—	—	—	
Winzes		.42	—	3.81	—	52.53	—
Grand Averages, all driving and timbering		1.71	2.35	6.36	6.91	61.53	40.28

The estimated cost of preparatory work per ton of ore to be mined is shown in Table 13.

Table 13. - Estimated cost of preparatory work for a block containing 120,000 tons, Morenci Branch, Phelps Dodge Corporation a

	Tons	Shifts	Units	Cost per unit				Total cost, labor, timber and explosives	
				Labor	Timber	Explosive	Total	Amount	Cost per ton
Supply drifts	600	87	130 ft.	$ 4.30	$ 2.20	$1.15	$ 7.65	$ 994.50	
Grizzly drifts	1,500	228	390 ft.	3.32	1.63	1.10	6.05	2,359.50	
Grizzlies	200	302	14 b	118.60	180.63	12.12	311.35	4,358.90	
Fingers	1,500	170	830 ft.	1.03	--	.89	1.92	1,593.60	
Pilot raises	200	54	80 ft.	3.14	.73	1.36	5.23	418.40	
Shrink drifts	500	78	160 ft.	2.59	—	1.07	3.66	585.60	
Totals	4,500	919	1,590 ft.					10,310.50	$ .086
Shrinkage stopes (tonnage used is swell only)	1,500	140	9,000sq.ft.	.10	—	.08	.18	1,620.00	
Undercutting (tonnage used is from dog holes only)	3,300	336	10,000sq.ft	.28	—	.22	.50	5,000.00	
Totals	4,800	476						6,620.00	.055
Total above grizzly level	9,300	1,395	1,590 ft.	—	—	—	—	16,930.50	141
Haulage drifts	2,500	286	330 ft.	5.86	1.99	1.42	9.27	3,059.10	
Loading chutes	300	119	7	73.04	104.70	5.02	182.76	1,379.32	
Haulage raises	1,300	207	600 ft.	1.56	.30	.72	2.58	1,548.00	
Total below grizzly level	4,100	612	930 ft.	—	—	—	—	5,886.42	.049
Total development exc. of shrinks & undercut	8,600	1,531	2,520 ft.	—	—	—	—	16,196.92	.135
Grand total	13,400	2,007	2,520 ft.	—	—	—	—	22,816.92	.190

a - Area = 10,000 square feet. Lift = 145 feet. Grizzly Drifts at 28-foot centers and Fingers at 14-foot by 18-foot, 6-inch centers. This estimate was originally made (April, 1928) to assist in planning operations for another mine of the Phelps Dodge Corporation.

b - 7 single and 7 double grizzlies.

NOTE: All costs per ton are calculated on a basis of 120,000 tons production, unless otherwise noted.

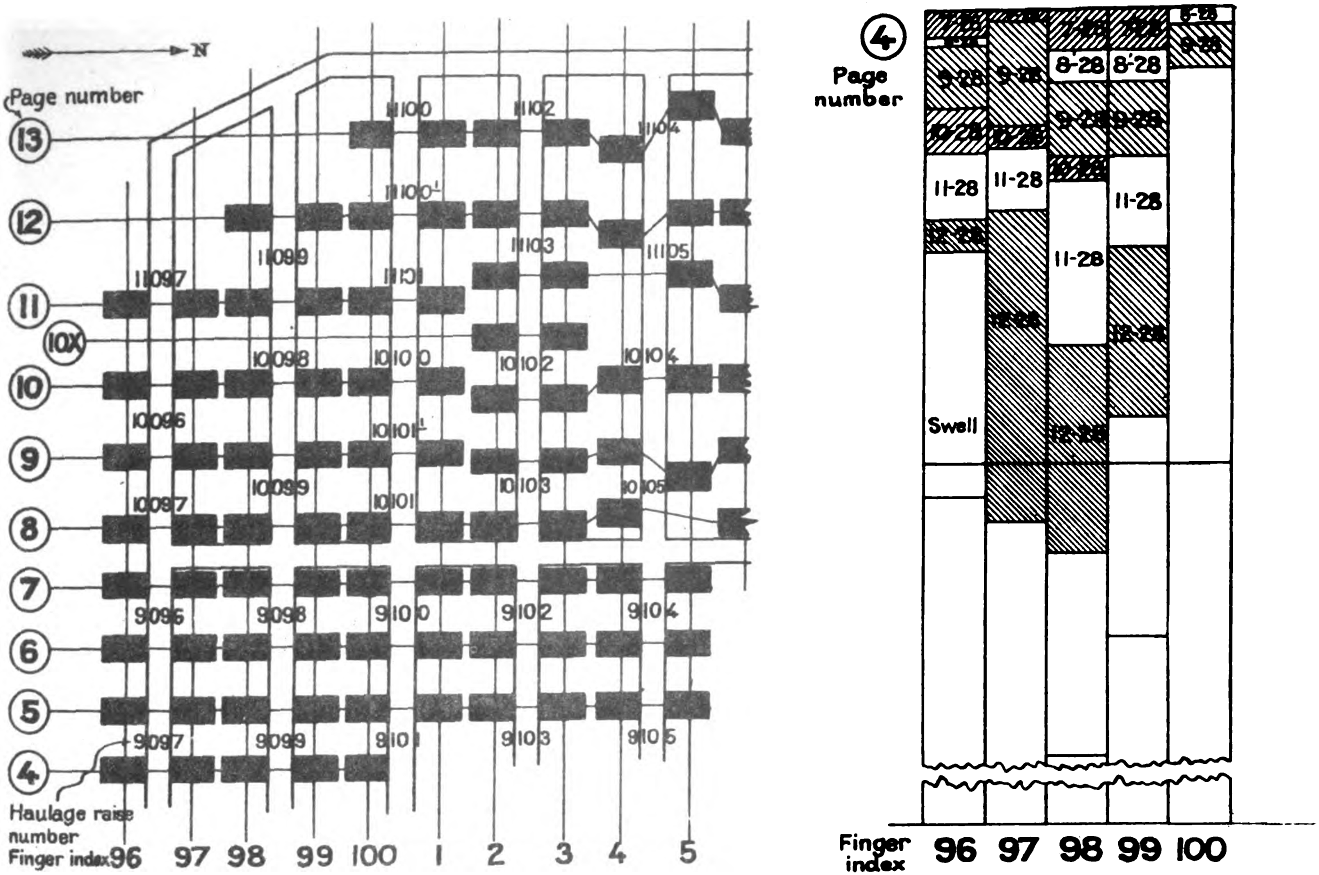
Table 13. - Estimated cost of preparatory work for a block containing 120,000 tons.

Morenci Branch, Phelps Dodge Corporation—Continued.

	Tramming		Hoisting		Sanitation		Ventilation		Assaying and sampling	
	Amount	Cost per ton	Amount	Cost per ton	Amount	Cost per ton	Amount	Cost per ton	Amount	Cost per ton
Supply drifts	$ 71.40		$33.40		$ 4.35		$13.92		$20.87	
Grizzly drifts	178.50		88.50		11.40		36.48		62.59	
Grizzlies	23.80		11.80		15.10		48.32		—	
Fingers	178.50		88.50		8.50		27.20		—	
Pilot raises	23.80		11.80		2.70		8.64		25.68	
Shrink drifts	59.50		29.50		3.90		12.48		25.68	
Total	535.50	$ .004	265.50	$ .002	45.95	$ .000	147.04	$.001	134.82	$.001
Shrinkage stopes (tonnage used is from swell only)	178.50		88.50		7.00		22.40		288.90	
Undercutting (tonnage used is from dog holes only)	392.70		194.70		16.80		53.76		—	
Total	571.20	$ .005	283.20	$ .002	23.80	$ .000	76.16	$.001	288.90	$ .002
Total above grizzly level	1,106.70	$ .009	548.70	$ .005	$69.75	$ .001	223.20	$ .002	423.72	$ .004
Haulage drifts	297.50		147.50		14.30		45.76		52.96	
Loading chutes	35.70		17.70		5.95		19.04		—	
Haulage raises	154.70		76.70		10.35		33.12		192.60	
Total below grizzly level	487.90	$ .004	241.90	$ .002	30.60	$ .000	97.92	$ .001	245.56	$ .002
Total development exclusive of shrinks and undercutting	1,023.40	$ .008	507.40	$ .004	76.55	$ .001	244.96	$ .002	380.38	$ .003
Grand total	1,594.60	$ .013	790.60	$ .007	100.35	$ .001	321.12	$ .003	669.28	$ .006

Table 13. - Estimated cost of preparatory work for a block containing 120,000 tons, Morenci Branch, Phelps Dodge Corporation—Concluded

	Drills and tools		Operating supplies		Mine department expense		Grand total	
	Amount	Cost per ton	Amount	Cost per ton	Amount	Cost per ton	Amount	Cost per ton
Supply drifts	$ 179.40		$14.30		$ 78.00			
Grizzly drifts	538.20		42.90		195.00			
Grizzlies	463.12		60.40		26.00			
Fingers	1,145.40		91.30		195.00			
Pilot raises	110.40		8.80		26.00			
Shrink drifts	220.80		17.60		65.00			
Total	2,657.32	$ .022	235.30	$ .002	585.00	$ .005	14,916.93	$ .124
Shrinkage stope (tonnage used is swell only)	1,100.40		143.00		195.00			
Undercutting (tonnage used is from dog holes only)	2,640.96		121.00		429.00			
Total	3,741.36	$ .031	264.00	$ .002	624.00	$ .005	12,492,62	$ .104
Total above grizzly level	6,398.68	.053	499.30	.004	1,209.00	.010	27,409.55	.228
Haulage drifts	455.40		36.30		325.00			
Loading chutes	77.98		23.80		39.00			
Haulage raises	828.00		66.00		169.00			
Total below grizzly level	1,361.38	$ .011	126.10	$ .001	533.00	$ .004	9,010.78	$ .075
Total development exclusive of shrinks and undercutting	4,018.70	$ .033	361.40	$ .003	1,118.00	$ .009	23,927.71	$ .199
Grand total	7,760.06	$ .064	625.40	$ .005	1,742.00	$ .014	36,420.33	$ .303



a *b*

Figure 35.– Stope control map of Morenci mine